



TRANSACTIONS
OF THE
AMERICAN INSTITUTE OF MINING
AND METALLURGICAL ENGINEERS
(INCORPORATED)

VOL. LXXV

CONTAINING PAPERS AND DISCUSSIONS PRESENTED AT MEETINGS
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[INCORPORATED]

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PREFACE

This constitutes the seventy-fifth volume of TRANSACTIONS published by this Institute. The first volume of the series covered the work of two years, 1871 and 1872, but so greatly has the work of the Institute expanded that as many as four volumes have been issued within a year. Collectively they make up the most complete compendium that has been printed of technical information covering mining and metallurgy through the years 1871 to 1927. For a number of years it has been impossible to include in the TRANSACTIONS all of the papers and discussions presented to the Institute. Much of the material is too highly specialized to be of interest to more than a minority of the members and a part, while highly important at the moment, is not suitable for inclusion in the permanent record. Preliminary studies, provisional reports of committees, descriptions of local plants and industries prepared to enrich the program of regional meetings, together with a large amount of what may be properly designated as news of the Institute, highly important to get to the members but again not TRANSACTIONS material, is necessarily now printed in some other form. This situation has come about gradually and to meet it the Board at first issued pamphlets, and special volumes. Later the *Bulletin* was established. Still later this was changed to MINING AND METALLURGY which now carries to all members the news of the organization, the reports of the meetings and many of the briefer technical papers, and an abstract of every paper printed by the Institute as well as a monthly Mining Index of papers on mining and metallurgy from all sources.

The highly specialized papers were formerly grouped together as Special Volumes and sold to members, and beginning in 1923 the annual review of the petroleum situation was issued separately but distributed without extra charge. For many years preprints of individual papers were made and distributed at the various meetings. Beginning in 1926 all the papers and discussions presented before the Petroleum Division were bound together in a separate volume, not one of the TRANSACTIONS, and sent to all members known to be interested in the subject or who asked for it. In 1927 a similar volume of PROCEEDINGS was established for the Institute of Metals Division and it is the hope of the Directors that the wealth of material available and the activities of other groups will permit the establishment and maintenance of still other series of special volumes as the years go by. Meanwhile the series of *Pamphlets* has been discontinued and instead the papers as

resented are printed as *Technical Publications*, properly grouped and numbered for reference.

It is proposed to make up the annual volume of TRANSACTIONS hereafter by selection from all of the material available in these forms of that portion which is of more general interest to the members and clearly of permanent worth. Where special volumes have been or are to be issued the Papers and Publications Committee is more restrictive in making its choice since the papers are or will be available to the members in bound form in some other volume. This policy explains the relatively large space given here to Iron and Steel, which has no special volume, as compared with Institute of Metals papers, where one has been issued. The Milling and Concentration papers are also, in the main, being withheld for inclusion in a separate volume to be issued within the year.

In order that this diversification in form of publication may not, in serving better the especial interest of each group, make difficult the keeping of an exact record of all the publications issued by the Institute from year to year, the publication of this volume of the TRANSACTIONS has purposely been held to the end of the year in order to include all titles. Beginning on page 873, accordingly, will be found a complete table of contents of the volume on PETROLEUM DEVELOPMENT AND TECHNOLOGY IN 1926, followed on page 879 by a similar table for the PROCEEDINGS OF THE INSTITUTE OF METALS DIVISION, 1927, and in turn on page 881 a complete classified list of all *Technical Publications*, with which has been included for convenience all *Pamphlets* issued before establishment of the series and since the beginning of 1921. The place of republication of each is indicated if it has been later included in the TRANSACTIONS or one of the special volumes. Finally the annual reports issued in the YEARBOOKS are cited and a complete classified index of the twelve numbers of MINING AND METALLURGY printed in 1927 is reproduced. With this information at hand any paper issued by the Institute in 1927, and in fact for the six years before, can easily be found. It is hoped that the convenience of this assemblage of information will be considered by the members to be sufficient compensation for the delay in issuing this volume.

H. FOSTER BAIN.
Secretary.

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HAROLD G. HIXON	R. M. ROOSEVELT	LELAND E. WEMPLE
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20. Dangers from Oil and Gas Wells to Coal Mining Operations

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21. Geophysical Methods of Prospecting

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New York Meeting

THE 135th meeting* of the American Institute of Mining and Metallurgical Engineers was held at New York, Feb. 14 to 17, 1927, with a total registration of 1493. It was the largest and most enthusiastic meeting of the Institute ever held.

Bradley Stoughton, Head of the Department of Metallurgy, Lehigh University, Bethlehem, Pa., delivered the Henry M. Howe Memorial lecture, the title of which was "Alloy Steels."

Prof. Cecil H. Desch, Sheffield University, England, delivered the annual lecture of the Institute of Metals. The title of his lecture was "The Growth of Metallic Crystals."

The technical sessions and round tables, 38 in number, were held over four days instead of three days as heretofore, in order to reduce the number of conflicts. Despite this arrangement as many as seven technical sessions were held simultaneously. Three special luncheons and five group dinners were held. Approximately 500 came together each day at the general luncheon and more than 700 attended the annual banquet. Four hundred and two attended the annual smoker at Mecca Temple.

The Petroleum Division held 14 technical sessions and gave a dinner at the Engineers Club. The Institute of Metals held five sessions, presented the lecture by Professor Desch and gave its annual dinner at the Fraternity Club. The Coal and Coke Committee held two sessions; the Mining Methods Committee one session; the Mining Geology Committee one session; the Mine Ventilation Committee three sessions; the Ground Movement and Subsidence Committee two sessions, and the Iron and Steel Committee several sessions, one of which was devoted to the Howe lecture presented by Mr. Stoughton. A round table on Carbon in Pig Iron resulted in so much pertinent discussion as to require two sessions. A number of papers were presented at a Mining and Milling session, and luncheons were arranged for various Committees and groups, among those getting together in this way being the Mining Methods, Milling Methods, Industrial Relations and Engineering Education Committees.

At the annual business meeting on Feb. 15, the following ticket was elected, and the reports of the President, Treasurer and Secretary were presented:

E. L. DeGolyer, Director and President; George Otis Smith, Director and Vice-President; F. Julius Fohs, Director and Vice-President; Frank H. Crockard, Director; David Levenger, Director; Richard Peters, Jr., Director.

* For full story of meeting see MINING AND METALLURGY (March, 1927) 106.

George Otis Smith was elected First Vice-President; C. F. Rand, Treasurer; H. Foster Bain, Secretary, and H. A. Maloney, Assistant Treasurer, at the meeting of the Directors on Tuesday afternoon.

The Section Delegates convened Monday morning and afternoon, William E. Kelly presiding at the first session and President E. DeGolyer at the second. Twenty-one delegates attended.

A joint conference of the Institute of Mining and Metallurgical Engineers and the Mining and Metallurgical Society of America was held Tuesday afternoon, Feb. 15, with George Otis Smith in the chair and 120 members of the two societies in attendances. The topic discussed was "The Exhaustion of Mineral Resources." A committee was appointed to prepare a definite plan of work.

The annual meeting of the Woman's Auxiliary was held Tuesday morning, Feb. 15, Mrs. George D. Barron presiding. Approximately 100 members were present. The program of entertainment for the ladies included shopping tours, luncheons, theatre parties, an informal dance and visits to various points of interest.

The annual reception and dinner was held at the Waldorf-Astoria on Wednesday evening, Thomas B. Stearns presiding as toastmaster. The James Douglas gold medal was presented to Zay Jeffries. The W. L. Saunders medal was presented to D. W. Brunton (who was represented on this occasion by John Wellington Finch). T. L. Joseph received the J. E. Johnson, Jr., award.

An opportunity was afforded the members to visit a number of the large plants in the metropolitan area, including the Holland vehicular tunnels, an important New Jersey oil refinery, a mechanical rubber goods manufactory, a 90,000-h.p. steam generating plant, a Western Electric Co. cable-making plant, the Eighth Ave. subway, etc.

TECHNICAL SESSIONS

Petroleum

PETROLEUM ECONOMICS SYMPOSIUM

JOSEPH E. POGUE, Chairman

The Place of Petroleum in Industry. ARTHUR KNAPP

The Economics of Gasoline. BARNABAS BRYAN, JR.

The Economics of Natural-gas Gasoline. G. R. HOPKINS

Economic Position of Fuel Oil. C. CAMPBELL OSBORN and C. J. DEEGAN

Economic Trend in Lubricating Oils. W. F. PARISH

The Oil Tanker Situation. J. V. BLAKE and C. P. WATSON

Distribution of Price Changes of Crude Oil. W. B. GEALY

ROUND TABLE DISCUSSION ON PETROLEUM ENGINEERING RESEARCH PROBLEMS

Led by WILLIAM E. WRATHER

REFINING SYMPOSIUM

CHARLES H. OSMOND, Chairman

The Chemistry of Cracked Gasoline. BENJAMIN T. BROOKS

Economic Aspects of Desulfurization in Refining Cracked Motor Fuels. JACQUE C. MORRELL and GUSTAV EGLOFF

Recent Developments in Fractionation as Applied to Tube Still Distillation. C. L. SMITH

Production of Lubricating Oils Using Fine Clays. E. R. LEDERER

Refining of Mexican Crude Petroleum. JOHN W. POOLE

Use of Automatic Control in Refining. LUIS DE FLOREZ

PRODUCTION ENGINEERING SYMPOSIUM

F. JULIUS FOHS, Chairman

Some New Aspects of the Gas Lift. E. O. BENNETT and K. C. SCLATER

Capillary Retention of Petroleum in Unconsolidated Sands. L. C. UREN and A. H. EL-DIFRAWI

Surface Tension Factors Developed in Flotation Research and Their Application to Flow of Crude Oil. A. W. FAHRENWALD

Some Problems of Foreign Petroleum Operations. CHESTER A. NARAMORE

Core Studies of the Second Sand of the Venango Group, from Oil City, Pa. CHARLES R. FETKE

JOHN M. LOVEJOY, Chairman

A Proposed Method of Oil Recovery by Combined Mining and Flooding. JOHN L. RICH

Advances in Natural Gasoline Manufacturing Methods. GEORGE P. BUNN

California Practice in the Transmission of Highly Viscous Oils through Pipe Lines. C. P. BOWIE

Transportation of Crude Oil. A. S. JONES

Selected Bibliography on Production Engineering and Transportation Engineering, 1925 and 1926. L. C. UREN

ROUND TABLE DISCUSSION ON PETROLEUM ENGINEERING PROBLEMS

Led by H. C. GEORGE

Production engineering, LESTER C. UREN, chairman; refinery engineering, FRED W. PADGETT, chairman; transportation engineering, L. C. LIGHTY, chairman; economics and management, ROSWELL H. JOHNSON, chairman; basic training and prerequisites, K. C. SCLATER, chairman; geological engineering, WILLIAM H. EMMONS, chairman; graduate courses, R. E. DAVIS, chairman; laboratories and equipment, A. F. MELCHER, chairman; availability of lecturers to supplement college staffs, CHARLES N. GOULD, chairman.

CORROSION SYMPOSIUM

FRANK N. SPELLER, Chairman

The Importance of Corrosion Problems. F. N. SPELLER

Corrosion in the Presence of Oily Substances. ULICK R. EVANS

Sulfur in Petroleum. A. E. WOOD

The Non-corrosive Ferrous Alloys. JOHN A. MATHEWS

Pump Corrosion. HAMILTON GARNSEY, JR.

Corrosion in Petroleum Industry Pumping Equipment. W. H. LAURY
 Corrosion of Underground Pipe Lines. E. P. BLY
 Soil-corrosion Investigation. K. H. LOGAN
 Scale and Corrosion Problems in Gasoline Plants. W. R. FINNEY and H. W. YOUNG
 Corrosion in the Refinery and the Use of Ammonia as a Means of Arresting Corrosion.
 D. E. PIERCE and WALTER SAMANS
 Corrosion in an Oil Refinery. H. F. PERKINS
 Corrosion of Pressure-still Equipment. GUSTAV EGLOFF and JACQUE C. MORRELL
 Corrosion in Oil and Gas Wells—Its Causes and Prevention. R. VAN A. MILLS

PRODUCTION SYMPOSIUM

JAMES H. GARDNER, Chairman

Petroleum Development in California during 1926. JOSEPH JENSEN
 The Los Angeles Basin. THOR WARNER
 Oil Resources of Kansas and Oklahoma in 1926. HENRY A. LEY
 Oil Produced in Texas during 1926 Exclusive of the Gulf Coast District. R. B.
 WHITEHEAD
 Operations in the Gulf Coast of Texas and Louisiana for 1926. SIDNEY A. JUDSON
 Review of Petroleum in Arkansas and Louisiana during 1926. H. D. EASTON
 Development in the Oil Industry in the Rocky Mountain Region during 1926. FRED
 E. WOOD
 Review of Appalachian Fields for 1926. GEORGE F. TURNBULL
 Petroleum Developments in the Mississippi Valley Region during 1926. GAIL F.
 MOULTON, W. N. LOGAN and R. LEE COLLINS
 Mexican Oil Fields during 1926. WALT M. SMALL
 Venezuela and the West Indies. A. FAISON DIXON
 Oil Development in Colombia in 1926. OLIVER B. HOPKINS
 Oil Development in Peru during 1926. OLIVER B. HOPKINS
 Oil Development in Ecuador during 1926. OLIVER B. HOPKINS
 Oil Development in Canada during 1926. OLIVER B. HOPKINS
 The World's Petroleum Production during 1926. VALENTIN R. GARFIAS
 Russian Oil Fields, 1925 to 1926. BASIL B. ZAVOICO

GENERAL SESSION—REVIEWS FOR 1926

F. JULIUS FOHS, Chairman

Review of Production of Petroleum in the United States in 1926. JAMES H. GARDNER
 Advances in Refining Technology during 1926. CHARLES H. OSMOND
 Advances in Drilling and Production Methods. ROBERT R. BOYD, WARD B. BLODGET
 and N. W. WICKERSHAM
 The Trend of the Petroleum Situation. JOSEPH E. POGUE

Institute of Metals Division

GEORGE C. STONE, Chairman

Preparation of Metallic Single Crystals and Twinning in Zinc and Zinc Single Crystals.
 O. E. ROMIG
 Plastic Deformation of a Zinc Single Crystal. SAMUEL L. HOYT
 Plastic Deformation of Coarse-grained Zinc. C. H. MATHEWSON and A. PHILLIPS
 The Solidus Line in the Lead-antimony System. E. E. SCHUMACHER and F. C. NIX

W. B. PRICE, Chairman

Note on the Distribution of Energy in Worked Metals and the Effect of Process Annealing Temperature on the Final Annealing Temperature of Fine Copper Wire. L. ZICKRICK and R. S. DEAN

Note on the Relation of Annealing Temperature to Conductivity of Copper Wire. J. C. BRADLEY

Beryllium-copper Alloys. WILLIAM H. BASSETT

Some Comparative Properties of Tough Pitch and Phosphorized Copper. W. R. WEBSTER, J. L. CHRISTIE and R. S. PRATT

A Study of the 470° C. Transition Point in Cast 60:40 Brass. FRANCES HURD CLARK
Study of Heat Treatment, Microstructure and Hardness of 60:40 Brass. FRANCES HURD CLARK

ZAY JEFFRIES, Chairman

Corrosion and Physical Properties of Some Alloys of Aluminum, Zinc and Tin. N. O. TAYLOR

Equilibrium Relations in Aluminum-manganese Alloys of High Purity. E. H. DIX and W. D. KEITH

Deformation of an Aluminum Alloy by a Constant Load. C. B. SADTLER and J. L. GREGG

Still Casting of Metals. P. H. G. DURVILLE

ANNUAL LECTURE

S. A. TAYLOR, Presiding

Growth of Metallic Crystals. PROFESSOR CECIL H. DESCH

ALBERT E. WHITE, Chairman

Plastic Deformation of Metals. J. T. NORTON and B. E. WARREN

General Theory of Metallic Hardening. R. S. DEAN and J. L. GREGG

Tungsten and Thoria. ZAY JEFFRIES and P. TARASOV

Grain Boundary Phenomena in Tungsten Filaments. E. S. DAVENPORT

JOINT SESSION OF INSTITUTE OF METALS DIVISION AND NON-FERROUS METALLURGICAL COMMITTEES

ARTHUR L. WALKER, Chairman

Acceleration of the Rate of Oxidation of Ferrous Iron in the Presence of Copper, and Its Application to the "Heap Leaching" Process. E. POSNJAK

Copper Alloy Systems with Variable Alpha Range and Their Use in the Hardening of Copper. M. G. CORSON

Notes on the Atomic Behavior of Hardenable Copper Alloys. E. C. BAIN

The Tarnish Resistance and Some Physical Properties of Silver Alloys. LOUIS JORDAN, L. H. GRENELL and H. K. HERSCHMAN

The Application of Microtome Methods to the Preparation of Soft Metals for Microscopic Examination. F. F. LUCAS

Coal and Coke

A. C. FIELDNER, Chairman

Discussions:

Use of Coal for Making Water Gas. Led by W. H. FULWEILER

Mining Coals for Gas Manufacture. Led by H. J. ROSE and D. J. DEMOREST
Adaptability of Various Coals as Generator Fuel in the Manufacture of Water Gas.

W. W. ODELL

Design Elements of a Coalotem Plant Carbonizing Coal at Low Temperatures. F. C. GREENE

FRANK HAAS, Chairman

Evaluation of Coals for By-product Coke Making. Report of Committee by C. E. LESHER

Fine-coal Cleaning by the Hydrotator Process. WALTER L. REMICK

Management in Mechanization. J. C. WHITE

Factors in the Ignition of Methane and Coal Dust by Explosives. G. ST. J. PERROTT

Mining Methods

GEORGE A. PACKARD, Chairman

Acceleration Stresses in Wire Hoisting Ropes. G. P. BOOMSLITER

Liquid Oxygen Explosives in Strip Coal Mining. G. B. HOLDERER

Hardness and Toughness of Rock. EMILE E. GYSS and HENRY G. DAVIS

Standard Maps for Iron Mines: Report of a special committee. FRANK BLACKWELL,

EMERSON D. MCNEIL and DONALD B. GILLIES

Mining Geology

E. F. BURCHARD, Chairman

Geology of Pioche, Nevada, and Vicinity. L. G. WESTGATE and ADOLPH KNOPF

Drainage in the Red Iron Ore Mines of the Birmingham District, Ala. W. R. CRANE

Magnesite Mining in California. LEROY A. PALMER

Clays of Southern California. ROBERT LINTON

Relations of the Disseminated Copper Ores in Porphyry to Igneous Intrusives. W. H. EMMONS

Geology of the Manganese Ore Deposits of the Gold Coast, Africa. A. E. KITSON.

With Notes on the Petrology of Certain Associated Manganese-silicate-bearing Rocks, by N. R. JUNNER

Collection of Elements at American Museum of Natural History. GEORGE F. KUNZ
Metallization from Basic Magmas. CARLTON D. HULIN*Mining and Milling*

R. M. RAYMOND, Chairman

Degree of Liberation of Minerals in Alabama Low-grade Red Iron Ores after Grinding. W. H. COGHILL

Beneficiation of Alabama Siliceous Red Hematite. JOSEPH T. SINGEWALD, JR.

Recent Developments in Classification. A. M. GAUDIN and W. L. REMICK

Underground Deep-hole Prospecting at the Eagle-Picher Mines. W. F. NETZEBAND
Facts about Mine-timber Preservation. GEORGE M. HUNT
Potential Source of Asphalt-base Hydrocarbons in Northern Alberta. S. C. ELLS.
Mine Finance. SAMUEL H. DOLBEAR

Mine Ventilation

GEORGE S. RICE, Chairman

Methane Content of Coal-mine Air. WILLIAM P. YANT
Devices for Detecting Dangerous Gases in Mine Air. J. T. RYAN

Discussions:

Cross-cuts or Break-throughs in Coal Mining. J. J. RUTLEDGE
U. S. Bureau of Mines Definition of the Gaseous Mine. FRANK HAAS and E. A. HOLBROOK
Proposed Standards of Ventilation in Coal Mines. E. A. HOLBROOK
The Occurrence of Fire Damp in Bituminous Coal Mines. FRANK HAAS

A. C. CALLEN, Chairman

Report of Committee on Physics of Ventilation. A. C. CALLEN
Report of Committee on Metal Mine Ventilation. D. HARRINGTON
Underground Air Conditions and Ventilation Methods at Tonopah, Nevada. B. O. PICKARD
Ventilation of the Liberty Tunnels at Pittsburgh. LOUIS F. HUBER
Use and Dangers of Booster and Auxiliary Fans as Applied to Coal Mining Ventilation. H. I. SMITH

R. R. SAYERS, Chairman

Physiological Effects of Mine Dust. DR. EDGAR L. COLLIS
Source of Dust in Coal Mines. J. J. FORBES and A. H. EMERY

Ground Movement and Subsidence

H. G. MOULTON, Chairman

What Duty to Support the Surface Does a Subsurface Owner Owe? ROBERT G. BOSWORTH

Discussions:

Digests of Report of Rand Rock Bursts Committee; led by GEORGE S. RICE
Outbursts in Coal Mines on Vancouver Island; led by ROBERT HENDERSON and R. R. WILSON

Subsidence and Earth Movements Caused by Oil Extraction, or by Drilling Oil and Gas Wells; led by W. T. THOM, JR.

Iron and Steel

JOHN A. MATHEWS, Chairman

Nature of the Chromium-iron-carbon Diagram. M. A. GROSSMANN
A Theory of the Cause of Blisters on Galvanized Sheets. L. B. LINDEMUTH
An Introduction to the Iron-chromium-nickel Alloys. E. C. BAIN and W. E. GRIFFITHS

Some Characteristics of Low-carbon Manganese Steel. V. N. KRIVOBOK, B. M. LARSEN, WILLIAM B. SKINKLE and WILLIAM C. MASTERS
A Comparison of the Effect of Cobalt and Nickel in Steel. FRANKLIN H. ALLISON, JR.

HOWE MEMORIAL LECTURE

E. L. DeGOLYER, Chairman

Alloy Steels. BRADLEY STOUGHTON

ROUND TABLE: CARBON IN PIG IRON

R. H. SWEETSER, Chairman

Need for Research in Foundry Pig Iron. RICHARD MOLDENKE
Carbon Characteristics of Copper-bearing Pig Iron. W. B. COLEMAN
A Pig Iron, Low in Total Carbon, Is in Demand for Use in Various Industries. E. TOUCEDA
Carbon in Pig Iron. RALPH H. SWEETSER

Cleveland Meeting

A two-day regional meeting,* under the auspices of the Ohio Section and the Iron and Steel Committee, was held at Cleveland, Ohio, April 19 and 20, about one-half of those in attendance being from the Cleveland district and the other half from a distance. The special subject of the meeting was "The Resources and Metallurgy of Manganese Ores." C. K. Leith and J. V. W. Reynders presided at the two technical sessions. A dinner was held at the Hotel Cleveland on the evening of April 19 and was well attended.

On the second day of the meeting a visit was made to the ore-docks of the Pennsylvania R. R. Co., and the open-hearth, Bessemer and rod-mill departments of the Newburgh plant of the American Steel & Wire Co. The group also inspected the open-hearth furnaces and the new merchant mill at the Corrigan-McKinney plant.

TECHNICAL SESSIONS

Some Political Aspects of the World Manganese Situation. By C. K. LEITH
The Importance of Manganese in the Steel Industry. By H. M. BOYLSTON
Reserves of Lake Superior Manganiferous Iron Ores. By CARL ZAPFFE
Minnesota Manganiferous Iron Ores in Relation to the Iron and Steel Industry.
By T. L. JOSEPH, E. P. BARRETT and C. E. WOOD
Manganese Resources in Relation to Domestic Consumption. By JOHN V. W. REYNDERS
Geology of the Manganese Ore Deposits of the Gold Coast, Africa. By SIR ALBERT E. KITSON. (Paper is accompanied by Notes on the Petrology of Certain Associated Manganese Silicate-bearing Rocks, by Major N. R. JUNNER.)
Manganese in Non-ferrous Alloys. By M. G. CORSON
Iron-manganese Alloys Low in Carbon. By SIR ROBERT HADFIELD

* For full story of meeting see MINING AND METALLURGY (May, 1927) 207.

Salt Lake City Meeting

A regional meeting* of the American Institute of Mining and Metallurgical Engineers was held at Salt Lake City, Utah, Aug. 22-26, jointly with the fifth annual meeting of the Western Division of the American Mining Congress, 140 A. I. M. E. members attending. The A. I. M. E. sessions began Aug. 23, with a non-technical session over which J. O. Elton presided. The purpose of this session was to present in a non-technical way the relation of flotation to mining, smelting and the State. G. H. Clevenger, chairman of the Milling Committee, presided over a session at which were presented technical papers on flotation. One hundred and eighty members and others attended this session. The program of technical papers was continued at a second session over which D. A. Lyon presided. Dr. Lyon was also the chairman of a round table discussion of mill balls and crushing, participated in by G. H. Clevenger, John Gross, S. R. Zimmerley, Roy Hatch, Allen Kissock, A. M. Gaudin, G. H. Ruggles, J. M. Hyde, W. B. Cramer, R. A. Pallanch, R. W. Diamond, B. S. Morrow and others.

The program included a tour over the Alpine Scenic highway and a visit to the blast-furnace plant at Iron-ton. The Bureau of Mines film, "The Story of Copper," was shown at a local theater and the staff of the Bureau of Mines Experiment Station and of the University of Utah were "at home" to visitors. The Woman's Auxiliary of the Utah Section of the A. I. M. E. gave a reception to the visiting ladies at the home of Governor and Mrs. Dern.

The papers presented at the A. I. M. E. sessions were as given below. They are not printed in this volume of TRANSACTIONS but are being reserved for a special volume planned to be issued in 1928.

J. O. ELTON, Chairman

Definition, Present Status and Future of Flotation. ERNEST GAYFORD
Flotation and the Utah-Delaware Mine. FRANK WARDLAW, JR.
Flotation and the Park-Utah Consolidated Mine. PAUL HUNT
Flotation and the Utah Copper Mine. E. E. BARKER
Flotation and Lead Smelting: Roasting and Sintering. W. H. EARDLEY
Flotation and Lead Smelting: Blast Furnace. R. A. WAGSTAFF
Flotation and Lead Smelting: Zinc and Fluxes. A. B. YOUNG
Effect of Selective Flotation on Copper Smelting. R. W. SENGER
How Flotation Has Broadened the Geologist's Viewpoint. PAUL BILLINGSLEY
Economic Effect of Selective Flotation in Utah. W. MONT FERRY

* For full story of meeting see MINING AND METALLURGY (October, 1927) 422.

G. H. CLEVENGER, D. A. LYON, Chairmen

Flotation of Custom Lead-zinc-iron Ores as Practiced by the International Smelting Co. at Tooele. W. J. McKENNA

The Role of Sulfitcs in the Differential Flotation Plants of the U. S. Smelting Refining & Mining Co. R. A. PALLANCH

Development of Selective Flotation at Combined Metals Reduction Co., Bauer, Utah. R. J. EVANS, JR.

Flotation Mechanism, A Discussion of the Functions of Flotation Reagents. By A. M. GAUDIN

Ore Concentration Practice of the Consolidated Mining & Smelting Co. of Canada, Ltd. R. W. DIAMOND

Concentration of Lead-silver Ore at Hecla Mine, Gem, Idaho. W. L. ZEIGLER
Galena Flotation Concentrator, Lake Gulch, Idaho. W. L. ZEIGLER

Recent Flotation Practice at Inspiration, Arizona. GUY H. RUGGLES and HENRY F. ADAMS

The Forrester Cell Installation at the Nevada Consolidated Copper Co.'s McGill Concentrator. E. H. MOHR

Effect of Preferential Flotation at Cananea Mill and Smelter. A. T. TYE

Flotation Practice in the Coeur d'Alene District, Idaho. A. W. FAHRENWALD

Institute of Metals Division Meeting

THE Institute of Metals Division held its fall meeting* at Detroit, Mich., Sept. 19-24, in coöperation with the American Society for Steel Treating. The attendance for the first day of the Division's sessions was 105. Approximately 100 attended the Division dinner at which E. Blough, technical director of the Aluminum Co. of America, discussed the problems and progress of the aluminum industry.

The papers presented are given below. They are reserved for publication in a succeeding volume of the PROCEEDINGS of the Institute of Metals Division.

DR. ZAY JEFFRIES, Chairman

Condition of Thorium in Thoriated Tungsten Filament. ANCEL ST. JOHN
X-ray Analysis of Plastic Deformation of Zinc. T. A. WILSON and S. L. HOYT
Quantitative Spectrum Analysis. F. TWYMAN and D. M. SMITH
The Production of Metallic Single Crystals. J. A. M. VAN LIEMPT
Twinning in Ferrite. L. W. MCKEEHAN

W. R. WEBSTER, Chairman

Commercial Forms and Applications of Aluminum and Aluminum Alloys. P. V. FARAGHER
Machining Aluminum. R. L. TEMPLIN
Physical Characteristics of Commercial Copper-zinc Alloys. W. H. BASSETT and C. H. DAVIS
Nickel and Monel Metal, with Especial Reference to Annealing. C. A. CRAWFORD
Wrought Zinc. C. S. TREWIN

DR. S. L. HOYT, Chairman

Some Aspects of the Commercial Manipulation of Aluminum. C. F. NAGEL, JR.
Equilibrium Relations in Aluminum-silicon and Aluminum-iron-silicon Alloys of High Purity. E. H. DIX, JR., and A. C. HEATH, JR.
Heat Treatment of Aluminum-silicon Alloys. R. S. ARCHER, L. W. KEMPF and D. B. HOBBS

* For full story of meeting see MINING AND METALLURGY (October, 1927) 440.

Petroleum Division Meeting

THE fall meeting* of the Petroleum Division was held at Fort Worth, Texas, Oct. 19-20. More than 300 registered the first day, members coming from as far away as Venezuela and the eastern states.

The meeting was held under the auspices of the Production Engineering section of the Division. E. O. Bennett, chairman of the Fort Worth Section of the A. I. M. E., served as general chairman, and J. B. Umpleby, vice-chairman of the Division, was in charge of the program with a special committee in charge of the technical sessions, of which A. C. Rubel was chairman.

President E. DeGolyer spoke briefly at a smoker held Wednesday evening, Oct. 19, at the Fort Worth Club, after which J. D. Collette, chairman of the Texas Division of the Mid-Continental Oil & Gas Corp., delivered an address. J. E. Pogue discussed the economic problems of the industry. The papers are reserved for publication, together with those to be read at the New York Meeting in February, 1928, in the next special volume of the Petroleum Division.

TECHNICAL SESSIONS

AIR-GAS LIFT

JOHN M. LOVEJOY, Chairman

Principles of the Air-lift as Applied to Oil Production. H. R. PIERCE and JAMES O. LEWIS

Effect of the Gas-lift on the Physical Properties of Oil. R. R. BRANDENTHALER
Handling Recirculated Gas. R. D. GIBBS and C. C. TAYLOR

Mechanical Installations of the Gas Lift in Texas Other Than Gulf Coast Region.
E. V. FORAN

Mechanical Installations for Air-gas Lift in the Gulf Coast Area. L. L. BRUNDRED
Mechanical Equipment of Air-gas Lifts in Oklahoma and Kansas Other Than Seminole.

REID W. BOND

Mechanical Installations for Gas-air Lifts in the Seminole Area. CLIFTON R. SWARTS
Mechanical Installations for Gas-lift Pumping as Practiced in the California Oil
Fields. H. C. MILLER

AIR-GAS LIFT AND GAS-OIL RATIOS

J. B. UMPLEBY, Chairman

Air-gas Lift Practice in the Seminole Field. S. F. SHAW

Air-gas Lift Practice in California. A. C. RUBEL

The "Gas Factor" as a Measure of Oil-production Efficiency. LESTER C. UREN

* For full story of meeting see MINING AND METALLURGY (November, 1927) 471.

Effect of the Gas Lift on the Gas Factor and on the Ultimate Production of an Oil Well. E. O. BENNETT

The Relation of Air-gas Lift to Gas-oil Ratios and the Bearing on Ultimate Production (Data from California). F. W. LAKE

USE OF ELECTRICITY IN OIL-FIELD OPERATIONS

A. C. RUBEL, Chairman

Use of Electricity in the Mid-Continent Field. D. L. JOHNSON

Use of Electricity for Oil-field Operations in Wyoming. A. W. PEAKE and F. O. PRIOR

Relative Advantages and Costs of Electric Power in Lease Operations. L. J. MURPHY

HANDLING CONGEALING OILS AND PARAFFIN

E. O. BENNETT, Chairman

Summary of Existing Information on Handling Congealing Oils and Paraffins. C. E. REISTLE, JR.

Handling Congealing Oil and Paraffin Problems and Their Solution in the Appalachian Fields. FRANK M. BREWSTER

Handling Congealing Oil and Paraffin Problems and Their Solution in the Wyoming Field. F. E. WOOD

Problems Encountered in Handling Panhandle Crude. WM. VICTOR VIETTI and WM. A. OBERLIN

Necrology

The following is a list of members who died in 1926. It is compiled from reports to the Secretary's office.

YEAR OF ELECTION	NAME	DATE OF DEATH	YEAR OF ELECTION	NAME	DATE OF DEATH
1888	AERTSEN, G.	Apr. 21	1885	LENTZ, W. O.	*Nov.
1914	ALEXANDER, WILLIAM H.	Nov. 15	1892	LUNDBOHM, H. GALMAR.	
1903	BACORN, F. W.	Mar. 18	1922	MACKELVIE, N. STEWART	June 14
1889	BAYLISS, R. T.	May 4	1894	MCDOWELL, J. C.	Mar. 1
1890	BERG, P. T.	May 14	1904	MCLEANAHAN, J. K., JR.	Nov. 21
1921	BOHM, O. H.	Mar. 11	1918	METZGER, J. P.	Sept. 13
1903	BOTLE, EMMET D.	Jan. 4	1914	MEYER, ALBERT R.	Apr. 22
1915	BROOKER, CHARLES FREDERICK.	Dec. 20	1886	MITCHELL, ROBERT.	June 15
1914	BROWN, THOMAS J.	May 8	1883	MUDD, SEBLEY W.	May 24
1920	BUTLER, JOHN.	Sept. 22	1886	NEUSTADTER, ARTHUR.	Aug. 12
1920	CADWELL, W. C.	Feb. 16	1920	NEVILLE, NEIL.	Nov. 11
1915	CALLOWAY, ALFRED W.	Sept. 13	1880	OGLEBAY, EARL W.	June 22
1874	CANFIELD, FRED A.	July 31	1889	OMROD, JOHN D.	Feb. 23
1881	CHARLETON, A. G.		1905	OSGOOD, J. C.	Jan. 4
1874	COLTON, CHARLES A.	Jan. 1	1891	PARKE, WILLIAM G.	Sept. 29
1888	CONNOR, S. B.		1881	PERKINS, H. C.	June 5
1916	DAHL, SIMBON K.	Dec. 6	1873	PLATER, JOHN E.	Feb. 6
1914	DIAZ, EMILIO.	*Feb. 19	1910	POLEEMUS, J. H.	Feb. 10
1902	DODGE, CLEVELAND H.	June 24	1917	POMMERANTZ, K.	June
1924	DUNSTONE, WILLIAM	Sept.	1899	PORTER, WILLIAM H.	Nov. 30
1911	DUTTON, CHARLES E.	May 17	1905	POTT, JOHN N.	Apr. 17
1920	ELLIOTT, GEORGE K.	Sept. 22	1917	RAMAGE, ALFRED HULL.	Apr. 16
1924	FINLEY, WILLIAM H.		1919	REED, STALKER E.	Aug. 31
1875	FOSTER, ERNEST LENEVE.	*Sept. 21	1922	RHODES, WILLARD F.	July 25
1892	GARVIN, JOHN M.	Apr.	1920	RICH, WILLIAM J.	Jan. 25
1886	GILLESPIE, T. A.	Jan. 27	1882	ROSE, WILLIAM W.	Dec.
1914	GREENWAY, JOHN CAMPBELL.	Jan. 19	1892	ROSSI, AUGUSTE J.	Sept. 18
1914	HAYDEN, W. H.	Jan.	1920	RUSH, FRED.	Jan. 23
1921	HERBERT, GEORGE T.	June 21	1905	SCOVILL, * H. H.	
1895	HERZIG, C. S.	Nov. 18	1900	SMALL, WARREN N.	July 20
1899	HOBSON, ROBERT.	Feb. 25	1884	SMITH, OBERLIN.	July 18
1924	HOTTINGER, ALWIN J.	Feb. 13	1873	SMOCK, JOHN C.	Apr. 21
1917	HOWE, JAMES V.		1914	SOHNLEIN, M. G. F.	July 28
1921	HULL, ELTON B.	Nov. 22	1918	TABER, M. N.	Apr. 13
1921	HUNTLEY, STERLING.	May 26	1896	THURSTON, E. COPPEE.	Apr. 29
1915	IRWIN, EUGENE FRANCIS.	Feb. 4	1921	TOUGH, FEED B.	Oct. 24
1888	JONES, W. LARIMER.	Nov. 25	1913	WARE, H. S.	Dec.
1923	KAMM, E. O.	Feb. 27	1914	WENTZ, DANIEL B.	Feb. 9
1919	KARRI-DAVIES, WALTER.	Dec.	1922	WHEELER, LENTILHON.	Aug. 5
1891	KEMP, JAMES F.	Nov. 17	1923	WILSON, JOHN BARWISE.	Jan. 26
1907	KOERTING, BERTHOLD.	Oct. 24	1891	WILSON, M. OREME.	April 1
1924	KREKEL, EDWARD C.		1913	WONG, WILLIAM A.	Sept.
1921	LELAND, STANFORD.	Jan.			

* Died in 1925; not included in necrology for 1925, *Trans.* (1926) 73, xxviii.

Biographies

Biographical sketches of members who died in 1926 were published
in MINING AND METALLURGY as follows:

NAME	DATE OF DEATH	ISSUE 1926	PAGE
GUILLIAEM ABERTSEN.....	April 20, 1926	June	267
R. T. BAYLISS.....	May 4, 1926	Aug.	358
OSCAR H. BOHM.....	March 11, 1926	June	268
EMMET D. BOYLE.....	Jan. 4, 1926	Feb.	89
CLEVELAND H. DODGE.....	June 24, 1926	Aug.	357
GEORGE K. ELLIOTT.....	Sept. 22, 1926	Nov.	499
JOHN CAMPBELL GREENWAY.....	Jan. 19, 1926	Feb.	88
JAMES V. HOWE.....	May 16, 1926	Nov.	498
STIRLING HUNTLEY.....	May 26, 1926	Sept.	401
ERIC OSCAR KAMM.....	Feb. 27, 1926	April	182
CHARLES R. KEHLER.....	April 28, 1926	June	267
JAMES FURMAN KEMP.....	Nov. 17, 1926	Dec.	546
STANFORD LELAND.....	Jan., 1926	Feb.	83
SEEBLEY W. MUDD.....	May 24, 1926	July	308
ARTHUR NEUSTAEDTER.....	Aug. 12, 1926	Sept.	401
EARL WILLIAMS OGLEBAY.....	June 22, 1926	Oct.	456
JAMES HIGBIE POLHEMUS.....	Feb. 10, 1926	March	143
JOHN N. POTT.....	April 17, 1926	June	268
F. B. F. RHODES.....	Oct. 1926	Dec.	545
WILLARD F. RHODES.....	July 25, 1926	Dec.	545
OBERLIN SMITH.....	July 18, 1926	Aug.	385
JOHN CONOVER SMOCK.....	April 21, 1926	June	268
E. COPPEE THURSTON.....	April 29, 1926	June	268
FRED B. TOUGH.....	Oct. 24, 1926	Dec.	546
DANIEL B. WENTZ.....	Feb. 9, 1926	March	143
1927			
WILLIAM HOMER ALEXANDER.....	Nov. 15, 1926	Jan.	37
CHARLES F. BROOKER.....	Dec. 20, 1926	Feb.	90
SIMEON K. DAHL.....	Dec. 6, 1926	Jan.	38
ERNEST LENEVE POSTER.....	Sept. 21, 1925	Feb.	90
CHARLES S. HERZIG.....	Nov. 17, 1926	Jan.	37
ELTON B. HULL.....	Nov. 22, 1926	Jan.	38
WILLIAM LOHRMER JONES.....	Nov. 25, 1926	Jan.	38
WALTER KARBI-DAVIES.....	Dec. 1926	March	154
BERTHOLD KOERTING.....	Oct. 24, 1926	June	281
EDWARD J. MCCOOL.....	Oct. 25, 1926	March	154
JOHN KING McLANAHAN.....	Dec. 1, 1926	Jan.	37
HENRY CLEVELAND PERKINS.....	June 5, 1926	Jan.	38
WILLIAM H. PORTER.....	Nov. 30, 1926	Jan.	37
STALKER E. REED.....	Aug. 31, 1926	Jan.	38
HARRY S. WARE.....	Dec. 8, 1926	Jan.	37

PAPERS

Underground Deep-hole Prospecting at the Eagle-Picher Mines

W. F. NETZEBAND,* PICHER, OKLA.

(New York Meeting, February, 1927)

THE zinc-lead mines of the Eagle-Picher Lead Co. are situated in the Tri-state district of Oklahoma-Missouri-Kansas. The company is operating, at the present time, seven mines in the brecciated deposits of the Picher, Okla., camp; two in the water-course ground of the Treece, Kans., camp; two in the open, boulder ground of the Crestline, Kans., camp; and one in the sheet ground of the Pierce City, Mo., camp. The accompanying map (Fig. 1) shows the relation of the rich narrow runs to the other orebodies of the district. The runs are designated by cross-hatching so as to distinguish them from narrow drifts that have been driven as prospect drifts or haulage drifts. Figs. 2, 3 and 4 are cross-sections showing the type of formation encountered in the operation of the drills.

The Picher camp is in the brecciated type of deposit with some water-course ground encountered, usually near the edge of the deposit. These deposits are flat and fairly uniform so that they can be mined systematically. However, scattered between these deposits are narrow runs that are usually very rich. Up to the present time, most of the prospecting has been done from the surface with churn drills, supplemented with some prospect drifts in the ground, once the mine is opened up. Prospect drifts are expensive, do not give much information except the presence or absence of ore, and are therefore not used to any great extent.

In the early days of the camp only the larger deposits were found and mined, with an occasional narrow run which was drilled out more or less by sheer luck. Later, more of these narrow rich runs were found and mined, until today many of the properties that were formerly considered worked out are being leased and re-drilled. It is very evident that many of these narrow runs, which are often only 20 ft. wide, can easily be missed by drilling from the surface unless very close drilling is practiced. This, of course, becomes very expensive as there are from 150 to 250 ft. of barren rock above the ore horizons and drilling costs from \$1.00 to \$1.50 per ft. In numerous cases there are several roughly parallel narrow runs or a narrow run that roughly parallels a larger deposit for some distance. Also there are many barren ribs in the brecciated deposits of from 20 to 100

* Geologist, Eagle-Picher Lead Co.

ft. thick, with rich ore on the other side. It is evident that in these cases a drill capable of drilling a hole up to 100 ft. in depth, horizontally, would prove very valuable.

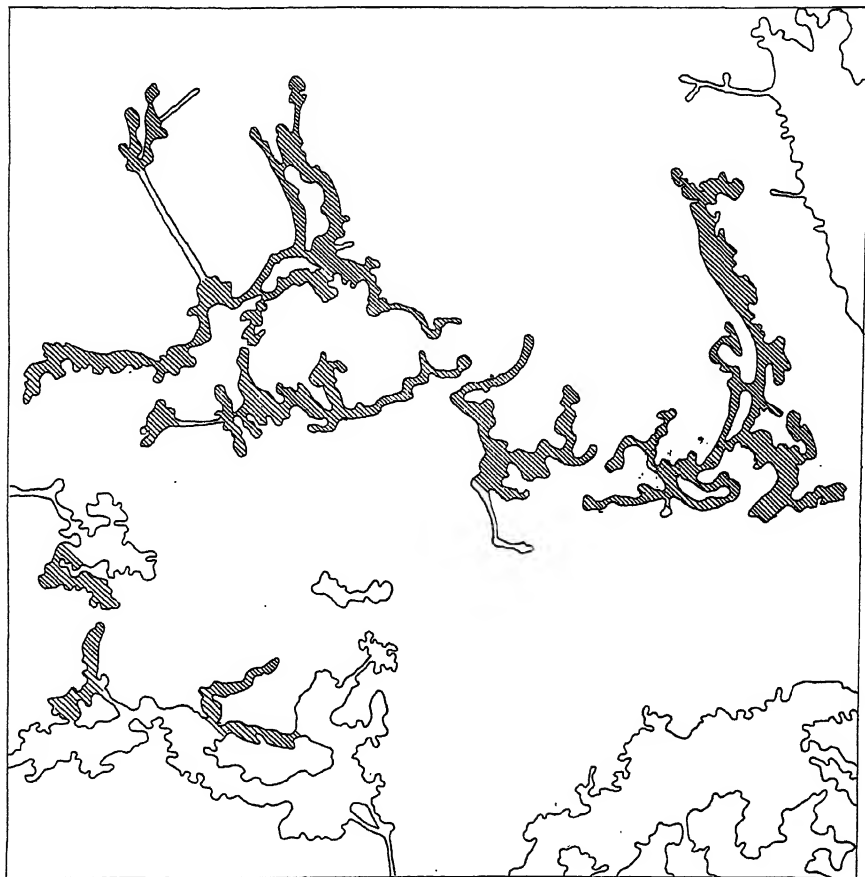


FIG. 1.—MAP SHOWING RELATION OF NARROW RUNS TO OTHER OREBODIES.

EQUIPMENT FOR DEEP-HOLE DRILLING

This type of drill has recently been introduced into the field by the Denver Rock Drill Manufacturing Co. in their model 34 "Waugh Turbo" drill, a heavy-duty, independently rotated drifter. Several of the other drill-manufacturing companies have introduced similar

machines into the district, but the writer is familiar only with the Denver machine.

The equipment, which makes the use of the machine possible for prospecting purposes, was perfected at the mines of the Chief Consol-

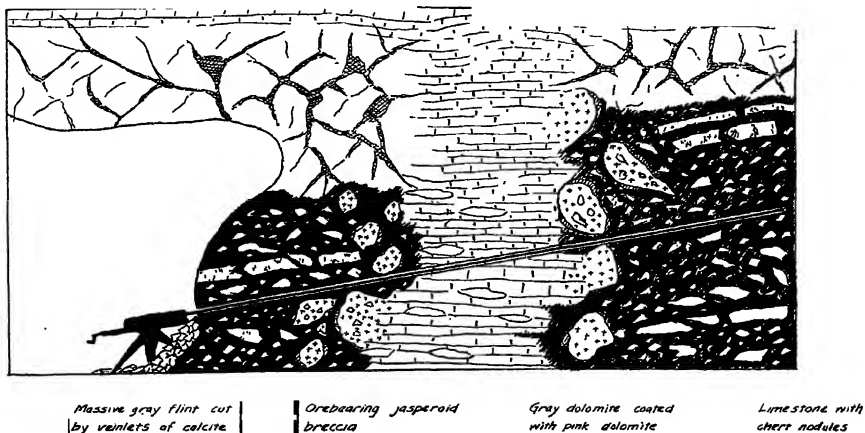


FIG. 2.—TYPICAL SECTION THROUGH LIMESTONE BAR SHOWING RELATION OF DOLOMITE TO OREBODIES.



FIG. 3.—TYPICAL SECTION SHOWING FRACTURING TRANSVERSE TO BEDDING OF MASSIVE CHERT AND RELATION TO OREBODY.

idated Mining Co., at Eureka, Utah, in cooperation with the Denver Rock Drill Manufacturing Co.¹ The joint of the steels together with the method of getting the water to the bit are two very important factors that were successfully overcome, and these are shown in Fig. 5.

¹ Chas. A. Dobbel: Deep-hole Prospecting at the Chief Consolidated Mines. *Trans.* (1925) 72, 677.

Bit

The bit used is the standard cross bit, which is the type used in the mining operations. Only two changes of steel are used. The starter is $3\frac{1}{4}$ -in. gage, this gage being used up to 50 ft. Beyond this a $3\frac{1}{8}$ -in. gage is used. This slows down the drilling speed on certain types of ground because some of the rock is very hard and wears away the gage very rapidly. In this type of ground dropping the gage would not help much unless the changes were frequent, and this would entail the use of a very

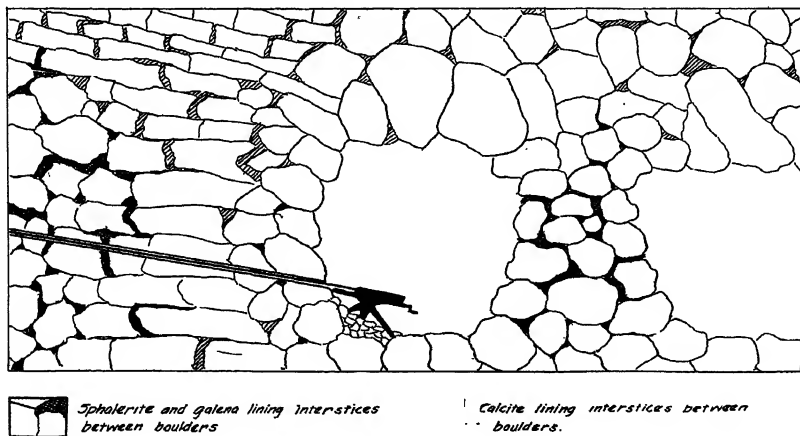


FIG. 4.—TYPICAL SECTION THROUGH BOULDER FORMATION.

large starter bit. Another factor that decided the use of only two changes of steel is the fact that this company uses only one machine of this type and the gages are larger than those used on the machines breaking ground. Using several changes of steel would require much extra labor for the blacksmith, besides the first cost of the dollies and dies, so that the actual saving is questionable in this particular case.

When hard ground is encountered and the loss of gage is excessive, reaming with the same sized gage is first tried and, if not successful, the hole is lightly squibbed. Squibbing is tried only when necessary as there is danger of pocketing the hole. These pockets are to be avoided whenever possible for the heavier ore minerals tend to settle in them and later may salt otherwise barren cuttings.

Operation

When the machine was first introduced into the ground it was placed in low headings, 8 to 12 ft. high, so that a double-post set-up could be used to advantage. Later, when the machine was moved into ground where

the drifts were cut high, sometimes as much as 70 ft., this set-up was found to be impracticable, and so a tripod was tried. It was necessary to get a heavy tripod to stand the excessive vibration. A "Sullivan U7" has been found to work very satisfactorily. It was at first thought that the excessive vibration of the heavy machine together with the vibration of the long steel would soon jar the tripod loose, requiring frequent resetting. Extra care was taken to sand in the legs of the tripod to give added firmness to the set-up, and it was found that then the tripod could be used without any trouble. Without the tripod the effectiveness of the machine

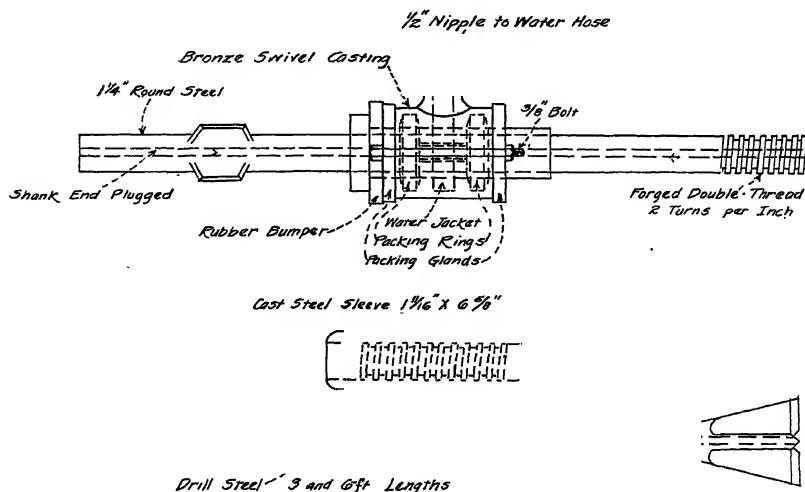


FIG. 5.—SWIVEL AND STEEL ASSEMBLY FOR DEEP-HOLE DRILLING WITH HAMMER DRILLS.

would be cut down considerably in this district. The set-up used is shown in Fig. 6.

Water is introduced into the steel through the swivel joint on the shank as shown in Fig. 5. Beyond 50 ft. a small 1-in. duplex pump is used to insure a strong flow of water to the bit. Some of the ground packs very readily and if a steel became "mucked in" in this type of ground, it would be difficult to recover, for it would sand in very rapidly. The water for the drill machines is under only the head acquired in bringing it from the surface, and as the mines are shallow (200 to 300 ft.), the water has not sufficient head to force itself very far into a hole.

USES OF THE DEEP-HOLE HAMMER DRILL

For deep drilling (depths up to 100 ft.), the machine is given a pitch of 10° above the horizontal. This makes for a very small difference in the

elevation of the collar and the back of the hole and at the same time gives enough slope to the hole so that the cuttings can be readily cleaned out. It is very often imperative that an upper level be prospected. In such cases vertical angles of from 20° to 30° are used. It has been found that angles of about 20° are the most satisfactory and that above 30° , too much of the weight of the steel is thrown back on the machine. This has resulted in excessive breakage of the swivel joint used for the introduction of the water.

The effectiveness of this machine is limited very definitely by the type of ground encountered. When open water-course ground is encountered,



FIG. 6.—SET-UP OF HEAVY DRIFTER USING TRIPOD.

it is practically useless as cuttings are unobtainable, and, of course, the cuttings are the important thing. In tight ground the machine has proved very effective, and although costs are still fairly high, it is a very valuable adjunct to the geological department.

METHODS OF SAMPLING

The sampling methods are much the same as those employed with the churn drills. A powder box is placed below the collar of the hole and all the cuttings for a 3-ft. screw are collected. The water is allowed to drain off and the cuttings thoroughly mixed. A grab sample of about 5 lb. is taken, sacked and sent on top to be examined and assayed if it is deemed necessary. The mine faces are not sampled in this district so that it is impossible to check the sampling of the drill holes. Sampling of the mine

faces is not imperative in this district for the values are fairly uniform and the mining can be done systematically. From checks on churn drill-hole sampling it has been found that reserves valued from churn drilling have milled out 10 per cent. higher than the drilling showed. It is thought that the sampling of the deep-hole prospect machine will compare favorably with that used for the churn drills.

COSTS OF DEEP-HOLE DRILLING

The costs of deep-hole drilling in the Tri-state district appear to be excessive when compared with the costs obtained by the Chief Consolidated mines at Eureka, Utah. There are several factors that may account for this. The Chief Consolidated are operating a number of these drills which makes for a lower maintenance cost, whereas the Eagle-Picher are operating a single machine. The character of the ground to be drilled is very different. The ground at the Chief Consolidated is limestone. The ground at the Eagle-Picher mines is flint and breccia with numerous openings. The flint is very hard, making for excessive wear of the bits, and the numerous openings cause considerable breakage that would not have to be contended with in tight ground. The harder ground cuts down the drilling speed to an average of 12 to 15 ft. per shift. In favorable ground 50 and 60 ft. have been drilled in one shift, but this is rather unusual, 25 ft. being considered a fair day's run.

The cost of deep-hole drilling shown in Table 1 is representative.

TABLE 1.—*Costs of Deep-hole Drilling*

	COST PER FT.
Labor.....	\$0.82
Maintenance labor.....	0.06
Repairs.....	0.99
Explosives.....	0.01
Power, oil and depreciation.....	0.24
	<hr/>
	\$2.12

Ore at Deep Levels in the Cripple Creek District, Colorado*

By G. F. LOUGHLIN,† WASHINGTON, D. C.

(Denver Meeting, September, 1926)

MORE than 20 years have passed since the publication of Lindgren and Ransome's report on the Cripple Creek District,¹ which was made when the district was much more active and prosperous than in recent years. They concluded that the output of the district, which had reached its maximum of about \$18,200,000 in 1900, would gradually decline, and their conclusion has been confirmed by records of subsequent production.

Between 1905 and 1915 annual declines were partly compensated by encouraging increases, especially in 1908, 1914, and 1915; but a rapid continuous decline followed, due largely to the war and its after-effects, until in 1922 the value of output was only slightly more than \$4,000,000. It rose to nearly \$5,000,000 in 1924, but declined again in 1925 and in 1926, when it was about \$4,400,000, and represented ores mainly from deep levels in the Cresson and Portland mines. The profitable operations of these two mines were the principal factors leading to a study of ore at deep levels in the district by the U. S. Geological Survey in cooperation with the Colorado Metal Mining Fund.

The principal reasons for Lindgren and Ransome's conclusion were three: (1) the greater cost and difficulty of deep exploration; (2) the decrease in the number of veins with increasing depth, and (3) the probability that the telluride ore, believed to have formed at comparatively low temperature, would not persist indefinitely in depth. Operations since they made these statements have verified the first two, but call for some modification of the third. Rich telluride ore is being mined at a profit in the Portland mine 3000 ft. below the present surface, or 5000 ft. below the original surface of the Cripple Creek volcano, as interpreted by Lindgren and Ransome. The question arises, therefore, as to the number and extent of such deep deposits, and the answer, so far as it can now be given, must be based mainly on structural data, supplemented by data on the processes of ore deposition. Presentation of these data based on a study of deep mines in the southeast quarter of the district, the only

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† Geologist in Charge, Section of Metalliferous Deposits, U. S. Geological Survey.

¹ W. Lindgren and F. L. Ransome: *Geology and Gold Deposits of the Cripple Creek District, Colo.* U. S. Geol. Surv. *Prof. Paper* 54 (1906).

part covered by the writer thus far, is the object of this paper which is presented as a supplement to Lindgren and Ransome's report and mainly for those especially interested in the district. Incidentally, brief attention will be given to a few interesting details that have been discovered since the publication of Lindgren and Ransome's report. These are the local pipe of basaltic breccia at the Cresson mine and the low-grade pyritic ore with its local concentration of cinnabar in collapse breccia at the Dante mine.

SUMMARY OF GEOLOGY

Main Volcanic Area

The geology of the district has been thoroughly described by Lindgren and Ransome, and the reader is referred to their report for anything more than the very brief outline that follows:

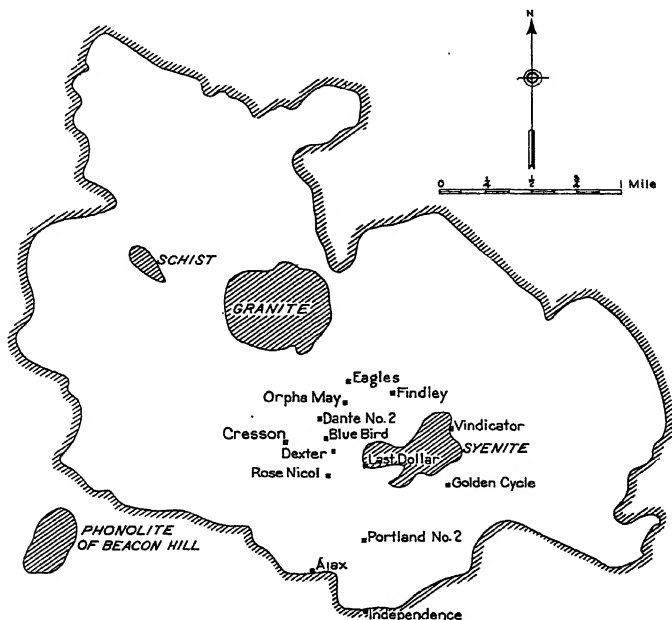


FIG. 1.—OUTLINE MAP OF CRIPPLE CREEK VOLCANIC AREA, SHOWING LOCATION OF MINES MENTIONED IN THIS PAPER.

The mines of the district are mostly situated within a complex volcanic neck of phonolite breccia about 4 miles in major diameter, which is outlined in Fig. 1; but a few important mines are in the surrounding Pre-Cambrian granite, especially near the south margin of the neck in

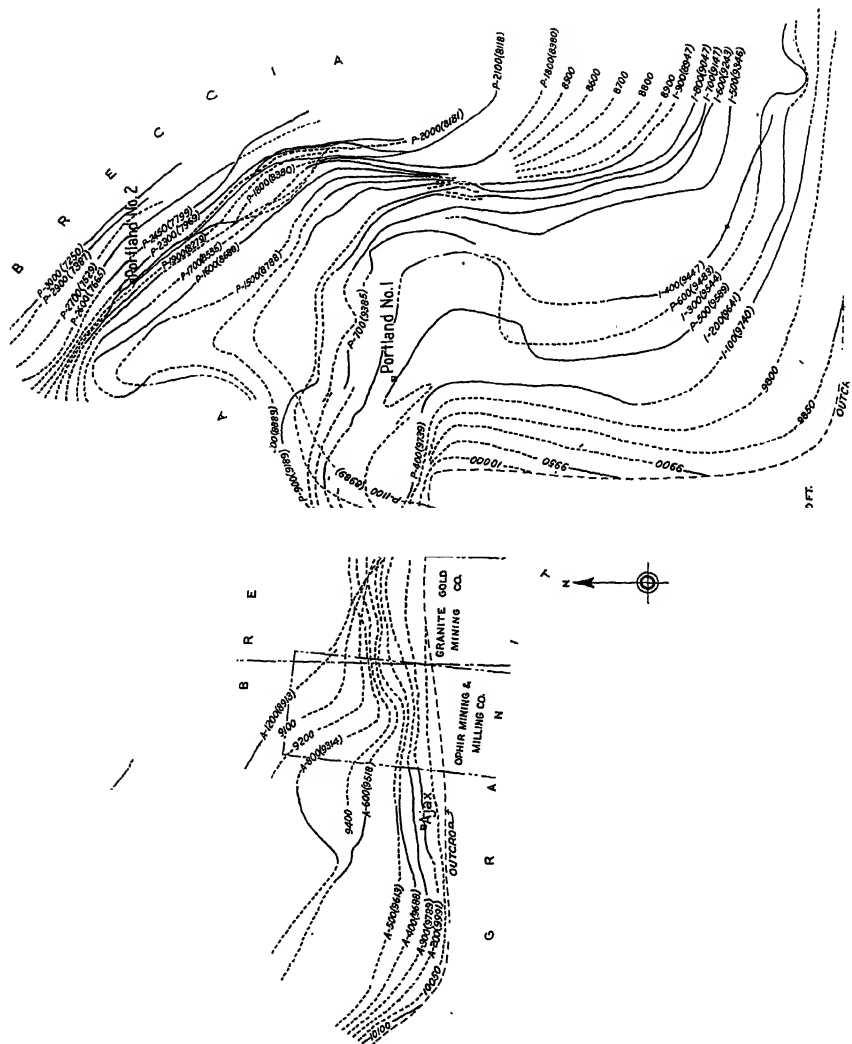


FIG. 2.—PLAN OF GRANITE-BRECCIA CONTACT ON DIFFERENT LEVELS OF THE AJAX, PORTLAND AND INDEPENDENCE MINES.

Solid lines represent actual exposures and to a minor extent intervals between closely spaced exposures. Dashed lines are interpolations, and between the Ajax and Portland mines are mostly copied from Plate 5 of *Professional Paper 54*. Outcrop of the contact is copied from Plate 2 of *Professional Paper 54*.

the vicinity of Victor and around the outlying phonolite plug of Beacon Hill on the southwest margin of the neck. The walls of the neck are steep and rather irregular. They exhibit local bench-like flattening and in some places overhang the breccia, as shown in Fig. 2. Exposures of the walls underground are most abundant near Victor in the Ajax, Portland and Independence mines, and give the impression that on the whole the neck narrows downward. Underground exposures of the contact elsewhere are too few to outline it with satisfaction, but on the whole give a similar impression.

The neck incloses in its central and western parts "islands" of Pre-Cambrian granite and schist. The granite "island" has been proved by mine developments since Lindgren and Ransome's survey to be continuous with the main area of granite and schist to the north and probably with schist to the west.² This evidence together with the steepness of the few exposed contacts indicate that the northwest quarter of the breccia area is in reality a minor neck, as suggested by Lindgren and Ransome.³ The prevalence of intrusive masses in the breccia south and east of the granite "island" indicate that igneous activity was greatest there. This part of the breccia area also includes the deepest and most productive mines, and appears in the light of recent mining developments to be the main neck of the complex. The breccia in this southeast part is continuous, however, at least to the depth of the Roosevelt drainage tunnel, with that in the southwest part, where no recent mining developments have been studied. The phonolite plug of Beacon Hill marks a minor center of volcanic activity just outside of the southwest edge of the area.

The intrusive masses that cut the breccia include in order of abundance phonolite, latite-phonolite, syenite and alkaline basaltic rocks. Latite-phonolite, which includes at least two varieties, is the oldest and forms dikes, irregular downward-tapering plugs, and sill-like masses. The plugs and sill-like masses cover much of the surface in the southeast quarter of the area and are cut by the upper workings of several mines, but only dikes are cut by the lower levels. Dark gray syenite occurs principally as a plug or a closely-spaced group of plugs that forms a roughly elliptical area on the surface between the Last Dollar and Vindicator mines (Fig. 1) and a smaller circular plug a little farther west. These plugs have been cut at different places as far down as the Roosevelt drainage tunnel, 2100 ft. below the surface in the Portland and Vindicator mines; but the boundaries between syenite and breccia have not yet been adequately mapped except in the Vindicator mine, where the syenite forms a closely-spaced series of small plugs and dikes instead of one con-

² Oral information by C. O. Moss, engineer-geologist, Stratton Leasing Co.

³ W. Lindgren and F. L. Ransome: *Op. cit.*, 3.

tinuous plug as it appears to be on the surface. Dikes of syenite are found in mine workings near the plugs and as far north as the Eagles mine, but have not been separately mapped on the surface.

Phonolite includes a common, light gray and a less common dark purplish to brown variety, the relative ages of which have not been determined. It forms a few plugs in the surrounding granite, notably at Beacon Hill (Fig. 1), but in the area studied by the writer it occurs only as dikes cutting all other kinds of rocks except basaltic dikes.

The basaltic dikes include trachydolerite, vogesite and monchiquite, but their distinction from one another is not important, unless it is desired to prospect along a certain dike on different levels; then it is their megascopic character such as the presence of large scattered biotite crystals, round white spots of analcite, spots and parallel streaks of calcite, or certain other kinds of alteration that are helpful rather than the microscopic distinctions on which their names depend. In the Cresson mine basaltic and other dikes are cut by a pipe of basaltic breccia, which is described below.

No one kind of rock has been especially favorable for the deposition of ore, as ore shoots have been found in or close by several different kinds. There has been a tendency to regard certain rocks with favor because of local conditions. Thus a phonolite dike practically coincides for much of its length and depth with the main vein in the Portland mine; latite-phonolite is mineralized in the outlying parts of the Cresson mine; a narrow basaltic dike (the "Sliver dike") in the same mine has been regarded with favor, whereas a larger basaltic dike (the "Funeral dike") only a few feet west of it has been regarded with disfavor, although it contained an ore shoot at one place; some of the veins in the Eagles mine extend along dikes of dark gray syenite. Other veins in many different places are walled by breccia or by granite. Some kinds of rock were doubtless more readily affected by vein-forming solutions than others, but reactions between rocks and solutions have had little to do with the localization of ore shoots. It was the reopening of certain dike fissures that permitted ore to form along the dikes, especially those with a marked platy or sheeted structure that originated during consolidation of the dikes.

Cresson Pipe of Basaltic Breccia

Although subsequent mining developments, as would be expected, have disclosed some dikes in addition to those mapped by Lindgren and Ransome, the only important discovery of an igneous rock is a pipe of basaltic breccia in the Cresson mine. This pipe, locally called the "Cresson blowout," is younger than all the dikes in the vicinity except a few short basaltic dikes that occur within it, and has had marked control

over the deposition of the larger orebodies of the Cresson mine. Its outline on the surface is elliptical, but it is there so thoroughly decomposed to brown soil that it doubtless was regarded for a long time as the ordinary breccia of the district with an original content of pyrite somewhat higher than ordinary. Its true character was not suspected nor identified until 1908, when in the upper levels of the mine, the abrupt cutting off of basaltic and other dikes by a dark colored breccia led to the explora-

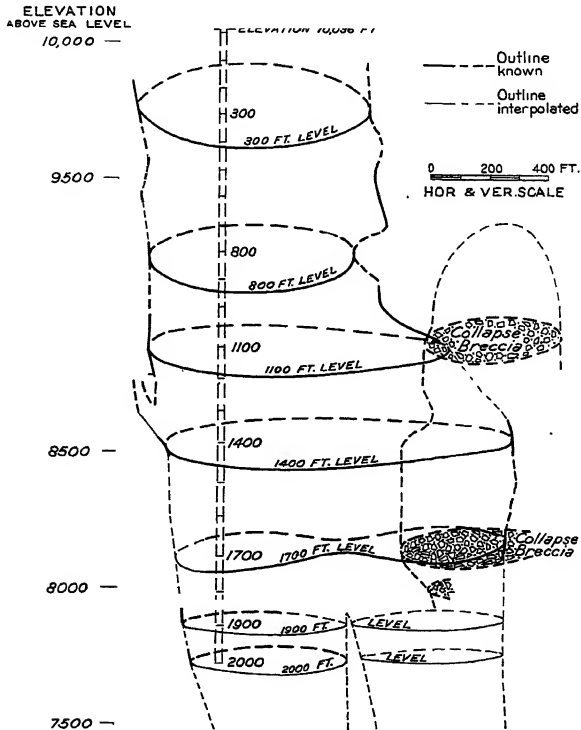


FIG. 3.—OUTLINE OF CRESSON PIPE OF BASALTIC BRECCIA.

tion and outlining of the local pipe. Its horizontal cross-sections on the upper levels are elliptical with major axes about 700 ft. long, and minor axes 500 ft. long (Fig. 3). Below the eighth level, it becomes more elongate, though not appreciably narrower, but at the seventeenth level it has increased to 1100 ft. in length and has narrowed to a width of 300 ft. or less. It pitches steeply east northeast and dips steeply south-southeast. The lowest four levels so far as developed along the pipe show a narrowing of its middle part and a tendency to divide into two tapering roots, as indicated in Fig. 3.

The characteristic appearance of the breccia is a very dark gray matrix with inclusions of the surrounding light gray breccia, and of phonolitic and basaltic dike rocks. Microscopic examination of the matrix shows it to consist of isotropic basaltic glass well impregnated with fine-grained carbonate (ankerite) and pyrite. Black silicates have been destroyed by alteration, and the dark color is due mainly to the dustlike grains of pyrite. According to the accompanying chemical analysis of the matrix by R. C. Wells of the U. S. Geological Survey, pyrite amounts to more than 4 per cent. and carbonate to more than 18 per cent. The remainder, in spite of the presence of microscopic inclosed grains of the surrounding rocks, has the composition of a somewhat altered alkaline basalt.

CHEMICAL ANALYSIS OF BASALTIC BRECCIA, CRESSON MINE, CRIPPLE CREEK DISTRICT,
COLO. R. C. WELLS, ANALYST

	Insoluble in Acid, Per Cent.		Soluble in Acid, Per Cent.
Silica.....	46.78		0.70
Alumina.....	12.65		1.77
Ferric oxide.....	0.49		Not determined
Ferrous oxide.....	None		1.94
Magnesia.....	0.65		2.57
Lime.....	0.50		5.39
Soda.....	3.70	Manganese oxide.....	0.80
Potash.....	6.92	Carbon dioxide.....	8.30
Titanic oxide.....	0.74	Sulfur trioxide.....	0.41
Zirconia.....	0.08	Total soluble.....	21.16
Phosphorous pentoxide....	None	Total insoluble.....	77.42
Baryta.....	0.07	Water -	0.66
Sulfur.....	0.28	Water +	0.55
Arsenic.....	0.22		99.79
Chlorine.....	0.02		
Fluorine.....	None		
Tellurium and Selenium....	None		
Pyrite.....	4.32		
	77.42		

Fissuring

Fissuring within and around the main volcanic neck took place intermittently as shown by the cutting of older by younger dikes, and the cutting of all dikes by veins. Some were opened repeatedly and contain one or more dikes as well as vein material; others were opened during

only one or two stages, and if formed at an early stage remained sealed afterward. Many of the dikes of phonolite and basaltic rock as well as the principal veins fill fissures or sheeted zones that have a roughly radial arrangement, and converge toward centers that are distributed around a relatively barren area that includes the "islands" of granite and schist shown in Fig. 1; but many dikes and veins do not conform to such an arrangement, and several veins as well as barren fissures form distinct intersecting systems. Besides the systems of vertical and steeply dipping fissures, there are others called "flats" that are nearly horizontal or dip at angles of less than 45°. Both steeply dipping fissures and flats may contain veins, and ore shoots in one kind may end against the other. Lindgren and Ransome⁴ concluded that the fissures were caused by compression under a light load due to a slight settling of the consolidated breccia in the downward tapering neck.

The pipe of basaltic breccia in the Cresson mine affords a good example of fissuring due to slight settling. Both steeply dipping and "flat" fissures and sheeted zones are present. Those of steep dip in part coincide with the boundaries of the pipe, in part are roughly parallel to it, and in part are at right angles to it. A few trend in other directions. The "flats" are apparently most numerous in the outer parts of the pipe, but the interior has been less explored. Those in the southern or hanging-wall part of the pipe commonly dip toward the boundary, and those in the foot-wall part commonly dip away from the boundary. Short basaltic dikes locally fill the steeply dipping fissures within the pipe and along the boundaries, especially the south boundary. They less commonly fill "flats" but have formed large sill-like masses along them near the 1200 and 1800-ft. levels of the mine. Orebodies are largest and most continuous along the hanging-wall side of the pipe, and extend into the interior, especially near the sill-like mass on the 1200-ft. level, and into the surrounding ground along fissures that extend upward from the boundary of the pipe. The structure as a whole implies slight shrinkage of the mass accompanied by settling into the downward tapering pipe with a tendency to drag along the foot wall and to develop open fissures along and parallel to the hanging wall.

Fissuring in the main volcanic neck is similar in many respects to that within the Cresson pipe, but is much more complex. Irregularities in the walls of the neck and differences between the porous breccia and the dense latite-phonolite and syenite in resistance to compression and shearing have introduced many variations from the simple pattern of fissuring that would be expected from the settling of an inverted cone of uniform rock. In some places the arrangement of intersecting fissures and evidences of shearing suggest that

⁴ W. Lindgren and F. L. Ransome: *Op. cit.*, 167.

regional compression may also have been an important cause of fissuring; but when the complex composition and structure of the neck are fully considered, they appear adequate to account for the arrangement, although regional stresses may have supplemented local stresses. Productive fissures in the granite may have been formed during or before the earliest stages of volcanic activity, and have been reopened later, especially where they coincided in trend with fissures developed in the breccia, as in Ajax and Portland-Independence ground.

Interpretation of fissuring and the resulting distribution of productive and barren parts of veins is of utmost importance in the search for ore, but conditions affecting fissuring are so complicated that conclusions drawn from evidence in one part of the district may not apply in another part. The writer in this paper offers interpretations of local fissuring to account for the distribution of ore shoots in parts of the southeast quarter of the district, where data are comparatively abundant.

KINDS OF ORE DEPOSITS

Ore bodies were classified by Lindgren and Ransome as (1) fillings of simple fissures and sheeted zones in breccia or granite, or along dikes; and (2) irregular bodies adjacent to fissures and formed by replacement and crystallization of country rock, usually granite.⁵ Most of the ore bodies seen by the writer belong to the first class, but the large ore bodies in the Cresson mine belong to the second class, although the ore and other introduced minerals have replaced the matrix of basaltic and phonolitic breccia, either along the margins of the local pipe or within it. Another kind of deposit, exposed since Lindgren and Ransome's survey, consists of collapse breccia with fragments coated by vein minerals but thus far of too low grade to pay for mining. The best exposure of this kind is in the Dante and Cresson mines, and is described in some detail on page 53.

The mineral composition of the ores at deep levels is essentially identical with that of the original ore at the higher levels, and the relation of the minerals to one another is similar also; but the following features of the deposits are worthy of attention: stages of mineral deposition as an aid in development work; the occurrence of pyritic ore rich in gold; the occurrence and significance of low-grade ore in collapse breccia, and the occurrence of cinnabar in this low-grade ore.

Stages of Mineral Deposition

The order of mineral deposition is conveniently grouped for practical purposes into three stages, the second of which brought gold in commercial quantity. It is not necessary to enumerate the minerals of the

⁵ W. Lindgren and F. L. Ransome: *Op. cit.*, 153.

district here, but to emphasize significant associations. Dense masses of adularia and quartz are among the earliest minerals deposited, but the most conspicuous product of the early stage is dark purple, fine-grained, massive fluorspar accompanied by varying quantities of quartz and comparatively coarse-grained pyrite. The quartz is abundant but not conspicuous as most of it is very fine-grained or microscopic and is concealed in the dark, massive fluorspar. Its high percentage ruins the fluorspar for commercial use. This mixture of massive fluorspar and inconspicuous quartz commonly forms solid veins ranging from an inch or less to 1 or 2 ft. in thickness.

Locally the veins contain angular inclusions of wall rock and their middle portions here and there contain vugs in which minerals of the later stages may be present. Where split or broken by movement along the vein and especially at intersections with other veins or fissures they are likely to be filled with minerals of the second stage and to become valuable ore shoots, but elsewhere they are of no value. If operators can determine where such reopened places are to be expected, prospecting will be simplified, but where this cannot be done, the barren fluorspar vein must be followed for an indefinite distance in the hope that an occasional pay shoot will be found.

Minerals of the second stage include the same minerals as the first, but the fluorspar is usually somewhat lighter purple, the quartz milky to somewhat smoky, and the pyrite fine-grained and inconspicuous. As these minerals occur mostly in open though narrow cracks and in vugs, their crystal outlines are distinct. Other conspicuous minerals of this stage are dolomite in small, white, rhombic crystals, celestite usually in small slender prisms, and the tellurides, which form crystals in cracks and vugs and in some places also cut or impregnate the adjacent wall rock or the fluorspar and pyrite of the early stage (Fig. 4). An occasional grain or wire of free gold accompanies the tellurides. The green vanadium mica, roscoelite, also belongs to this stage, and is found occasionally as small, soft, drusy masses, but usually as a green coloring matter in rock along the edge of the vein or in inclusions. According to the writer's acquaintance with the district, which is limited to the mines mentioned in this paper, roscoelite is more conspicuous in basaltic than in other rocks. Robert Gardiner, superintendent of the Cresson mine, regards the green roscoelite stain and the delicate celestite crystals as especially favorable indications of good ore. Small quantities of sulfides of base metals belong mainly to this stage, and are commonly found near pay shoots.

The third stage, which includes two or more substages, is represented mainly by smoky to colorless quartz in distinct small to large, drusy crystals, and by druses of yellow chalcedony. Other minerals of this stage are pyrite, some of it in thin druses of radiating needles, and some

FIG. 4.

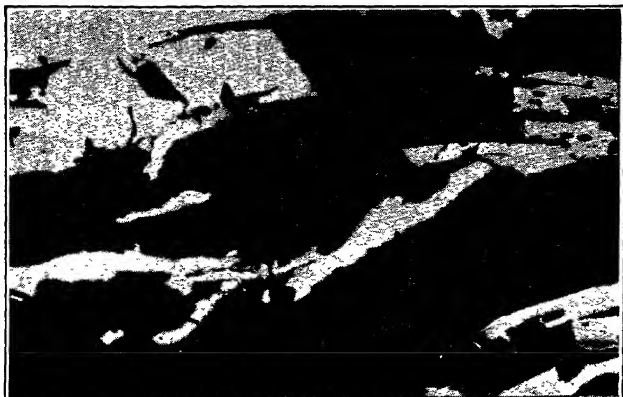


FIG. 5.

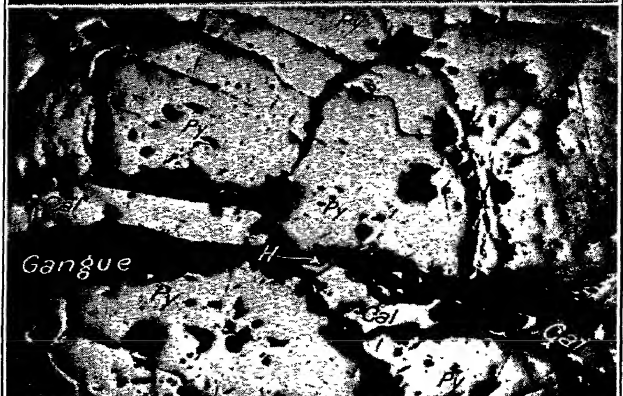


FIG. 6.



FIG. 4.—PHOTOMICROGRAPH OF A FINE-GRAINED MIXTURE OF FLUOR QUARTZ OF THE FIRST STAGE CUT BY CALAVERITE.

FIGS. 5 AND 6.—PHOTOMICROGRAPHS OF PYRITE (*Py*) OF FIRST STAGE CUTLETS OF GANGUE (DARK), CALAVERITE (*Cal*), AND HESSITE (*H*).

in small, fine-grained, drusy patches of 12-side crystals (pyritohedrons), calcite in small pointed crystals (scalenohedrons), and locally cinnabar. Minute grains of fluorspar are rarely present. The important feature of this third stage is that its minerals may conceal tellurides of the second stage, but they are also present in barren places and cannot be regarded as so definite an indication of ore as the minerals of the second stage. Quartz of the third stage has at several places replaced celestite, dolomite, and calcite.

It is noteworthy that although quartz and pyrite belong to all stages, their appearances are not the same in all stages, and they should aid rather than confuse in hunting for ore. Other minerals may be present in more than one stage, but conspicuous in only one.

Rich Pyritic Ore

The pyrite of pyritic gold ore represents both early and late stages. A specimen from the 2900-ft. level of the Portland mine proves under the microscope to be pyrite of the first stage shattered and recemented by telluride, fluorspar and quartz of the second stage. Another specimen of apparently solid pyrite but unusually rich in gold was microscopically examined by M. N. Short of the U. S. Geological Survey, and found to contain veinlets of gangue with calaverite and hessite, the silver telluride (Figs. 5 and 6). The calaverite both megascopically and microscopically so closely resembles pyrite in color that it is easily overlooked without resort to chemical tests. Hessite according to Lindgren and Ransome⁶ was not known from Cripple Creek. Its presence as microscopic interstitial grains in calaverite evidently accounts for the relatively high silver content of the ore in the Portland mine.

Pyritic ore from below the 1700-ft. level of the Cresson mine contains small vugs lined with small patches of pyrite of the third stage which conceal thinly scattered, minute telluride crystals. Lindgren and Ransome⁷ discussed the supposed occurrence of gold in pyrite, and described and figured a specimen in which telluride was covered by a druse of radiating pyrite crystals. Their statement still holds true that gold in the Cripple Creek district is present only as telluride and as occasional grains of free gold.

Low-grade Ore in Collapse Breccia in the Dante Mine and Its Relation to Other Ores

The low-grade ore in collapse breccia is best exposed on the 1500-ft. level of the Dante claim, 1200 ft. below the surface. This body extends southwestward into the adjacent Cresson ground (Fig. 7). Another

⁶ W. Lindgren and F. L. Ransome: *Op. cit.*, 116.

⁷ W. Lindgren and F. L. Ransome: *Op. cit.*, 30.

body has been exposed but not developed in the Coriolanus claim at the north end of the 1800-ft. level of the Ajax mine. A third has been penetrated to the south of the Roosevelt Tunnel near the Cresson lateral, but could not be studied by the writer because of bad air.

The ore in the Dante claim is most exasperating in containing minerals of all three stages but little or no telluride in commercial quantity. Charles Becker, who has operated the property under lease, stated that

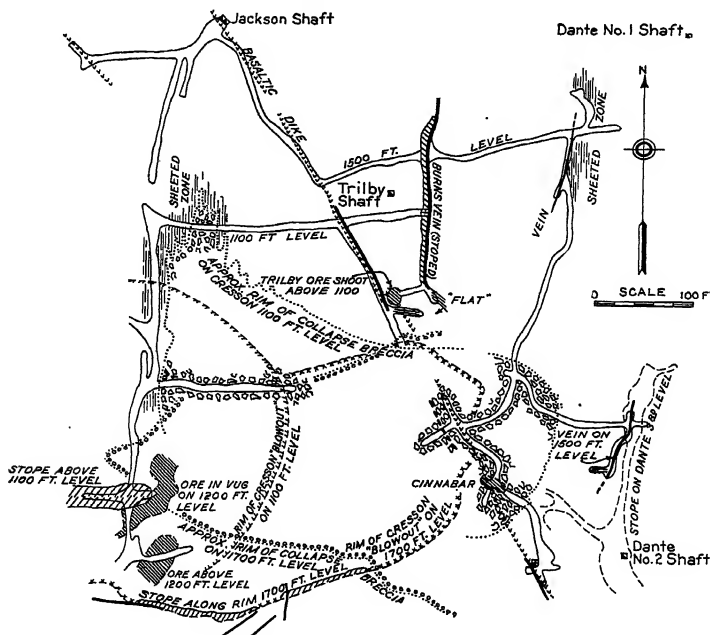


FIG. 7.—BODY OF COLLAPSE BRECCIA WITH LOCAL CONCENTRATION OF CINNABAR, EXPOSED ON THE 1500-FT. LEVEL OF THE DANTE MINE AND 1100 AND 1700-FT. LEVELS OF THE CRESSON MINE.

he found one specimen containing visible telluride, but that his regular samples assayed only \$3 or \$4 to the ton in gold. This ore consists mainly of fragments of country rock coated with minerals of the first and later stages, and fragments of first-stage fluorspar coated with minerals of later stages. Minerals of the second and third stages are also brecciated in turn. The extreme development of this material is a mass of subangular cobbles and small boulders of rock or dense fluorspar cemented mainly by quartz. These masses grade into rock with abundant "flat" sheeting and with considerable vertical sheeting.

The origin of the deposit is interpreted as follows: Early in the ore-forming period, solutions undersaturated with silica worked their way

upward, along vertical fissures and "flats" dissolving wall rock on the way. As cavities produced by corrosion left slabs of rock unsupported, they collapsed and their fragments were subjected to continuous corrosion. Where corrosion was most thorough, the original structure of the rock was completely destroyed, and a mass of rubble remained. Where corrosion was less thorough large vugs were formed along partly collapsed slabs, and where corrosion was least no collapse took place. Some vugs several feet in diameter were formed where solution was pronounced but the rock resisted collapse. The porous and much sheeted breccia was more readily attacked than were dense phonolite and basaltic dikes cutting it, although some large vugs were formed in basaltic rock.

This first stage of corrosion was followed by the first stage of mineral deposition, but collapse and shattering recurred intermittently and the resulting fragments were cemented by minerals of later stages. The latest shattering, of postmineral date, has left local bodies of uncemented fragments.

The process was doubtless more complicated than is indicated by this brief explanation. Further evidence may show that all neighboring veins of early fluorite preceded the shattering, as does the vein of *NNW* trend shown in Fig. 6. This relation may be explained by the inference that the very earliest solutions rising along trunk fissures had reached their saturation point and were forced to deposit at the level under consideration; that a continued supply of material during the later stages gradually allowed the solutions to rise higher and higher before becoming saturated, so that they were able to attack parts of the newly deposited veins. A change in composition of the solutions after the first stage may also account for the corrosion of the early fluorite veins.

This process as outlined is somewhat similar to that described by Locke⁸ in explaining the origin of the Pilares and other chimneylike masses of breccia cemented by ore and gangue minerals, but no well defined walls or marginal faults which he regards as characteristic have been found thus far in the inadequately exposed bodies at Cripple Creek. According to Locke's theory the fracturing is mainly due to collapse of the corroded rock under pressure; but in the Dante mine fracturing mainly preceded corrosion. Where chimneys of this kind have been bottomed they terminate in one or more small rootlike pipes.

LOCAL SOURCE OF ORES IN CRESSON AND NEIGHBORING MINES

The main significance of this kind of deposit, according to the interpretation offered, is that it lies above a local source of ore-forming solu-

⁸ A. Locke. Formation of Certain Orebodies by Mineralization Stopping. *Econ. Geol.* (1926) 21, 431.

tion. Solutions rose through the broken mass and escaped upward and through the sides, wherever opportunity offered. In the interior of the broken mass fragments became coated with gangue minerals, either because the saturation points of these minerals had been reached by decreasing temperature or because of reaction between solution and wall rock; but the more volatile or highly soluble tellurides were not so readily affected and continued upward through the continuous openings of the mass. The top of the brecciated mass has not been exposed but it would be reasonable to expect appreciable quantities of tellurides where the solution containing them became impounded either in the uppermost parts of the mass or along veins leading outward or upward from it.

The largest deposits in the vicinity of this brecciated mass, including those of the Cresson and Trilby, pitch toward it. The zones of weakness along and within the southern or hanging-wall part of the pipe of basaltic breccia in the Cresson mine afforded the greatest opportunity for ore deposition. Solutions rising westward along the upward pitch of the pipe's margin at first impregnated the basaltic breccia, partly replacing its matrix with gangue minerals of the second stage and rendering it more porous. The resulting structure permitted the solution to move slowly through the rock and to deposit tellurides in commercial quantity. The ore thus formed is very different in appearance from that in the ordinary veins, although both kinds of ore contain minerals typical of the second stage of deposition. Where intersecting fissures and shattering along the margin of the basaltic breccia permitted, ore impregnated the adjacent older breccia, and also formed pay shoots in typical veins near by. These veins, prominent on upper levels of the Cresson mine, pass downward into the mineralized breccia along the margin of the pipe.

Near the margin of the pipe on the Cresson's 1200-ft. level, a large cavity dissolved in a sill-like mass of basaltic rock served as a trap where solution could stagnate long enough to deposit the thick lining of telluride that created such excitement when discovered. Other large vugs have been found in the brecciated mass, but for one reason or another did not receive deposits of tellurides.

The Trilby ore shoot, now mined out, was a cylindrical body rising from near the margin of the collapse breccia a little above the 1500-ft. level of the Dante mine. Though it is inaccessible for detailed study its position is significant. The Blue Bird mine was idle during the writer's stay in the district, but its maps show that its veins have been stoped down to the 2100-ft. level, where they lie just east of the collapse breccia, and it is reported that mineralized "vuggy ground" or collapse breccia has been found in the bottom level. The Dexter and Hawkeye veins trend northward toward the body of collapse breccia exposed in the Dante mine, and the Dante vein, worked only down to the 300-ft. level, closely parallels its east side. The ore shoots as a whole in these veins pitch

toward the collapse breccia, and may well have been derived from the source beneath it.

The depth of this source can not be closely estimated. The orebodies in the Cresson and Blue Bird mines prove that ore can exist at levels well down along the sides of the mass of collapse breccia, and the prevailing pitch of ore shoots toward the mass may indicate the downward limit of ore.

Cinnabar in the Dante Mine

Had the tellurides in solution reacted readily with any of the other vein minerals or with wall rock their presence in the brecciated mass of the Dante mine would have been considerable. This was the case with cinnabar, which occurs almost entirely associated with pyrite. Where this relation is not evident the cinnabar forms small botryoidal druses in vugs, from which the solution which introduced the mercury evidently could not readily escape; but even some of these apparently independent druses are found, when broken open, to cover or be in close association with pyrite. One small specimen suggested that a crystal of zinc blende had been completely replaced by cinnabar.

Occasional spots of cinnabar are found throughout the brecciated mass on the Dante 1500-ft. level, but the mineral is most abundant within a very small area, and structural conditions there suggest that the mercury solution or vapor arose mainly where vertical openings afforded the most ready passage. Were cinnabar a common mineral in the district this occurrence would be a promising prospect. Under the circumstances its exploration by raising may disclose cinnabar in commercial quantity and will at the same time aid in the search for gold; but at present the cinnabar must be regarded mainly as an interesting mineralogical occurrence.

DISTRIBUTION AND LOCAL SOURCES OF ORE IN PORTLAND AND INDEPENDENCE VEINS

The Portland and Independence mines contain on their upper levels a group of roughly parallel veins of northward to northwestward trends more or less clearly linked together by cross-veins. Those of northward trend are close to the similarly trending part of the granite-breccia contact (Fig. 2) and cut either rock, and a few of them extend southward in the granite. As depth increases the veins either die out or coalesce into one vein of northward pitch that practically coincides with a phonolite dike along the granite-breccia contact. This vein has been followed to the 3000-ft. level which is the deepest in the district (altitude 7250 ft. above sea level). The orebodies on this lowest level are typical veins and there is no indication that their metal content is nearing its downward limit.

The distribution of pay shoots in the Portland veins and their southward continuation in the Independence mine has been governed by local causes of deformation, and an appreciation of these causes will greatly aid further developments in the vicinity. The principal local factors that controlled deformation are the granite-breccia contact and the two large masses of latite-phonolite exposed from the surface to the 1000-ft. level of the Portland mine.

The granite-breccia contact, as shown by many exposures in the Ajax, Portland and Independence mines (Fig. 2), is very irregular. As a whole, it extends eastward, with northward slope, from the Ajax shaft to near the Portland No. 1 shaft, where it turns and extends south for 1100 ft. and then extends eastward through the Independence shaft. On the deeper levels of the Ajax and on the 900 to 1100-ft. levels of the Portland mine west of No. 1 shaft, the few exposures available indicate a locally overhanging or southwestward dipping contact, which suggests either a local crater or the slipping of a large granite block downward to the north, accompanied by an unknown amount of shattering. The Bobtail vein which extends southeastward in breccia through the northeast part of the Ajax mine is parallel in strike to this overhanging contact, though its southwestward dip is probably less than that of the contact. It continues southeastward through the granite into Portland and Independence ground, where it has been productive on the upper levels. This vein has not been studied by the writer, but the suggestion is obvious that the Bobtail fissure in the granite prong between the masses of breccia was subject to reopening by local or regional disturbances, especially by the shearing along the main or contact fissure of the Portland mine which is considered below. A thorough study of such readjustments may determine whether deep prospecting along the Bobtail vein is to be encouraged.

East of its local overhanging part, the granite-breccia contact maintains its general direction of slope down to the 3000-ft. level of the Portland mine, though the degree of slope varies and small irregularities may be noted that suggest the slipping of slablike masses of granite into the crater. The most conspicuous of these is the "bench" between the 400 and 500-ft. levels of the Independence mine, northwest of the shaft. The turn from eastward to southward, which is rather sharp on the upper levels, becomes more gradual downward, and below the 1600-ft. level the contact has a southeastward instead of eastward course for a considerable distance.

The two latite-phonolite masses are exposed on the surface at and west of the Portland No. 2 shaft. They are very imperfectly outlined beneath the surface, but mine workings show them to be very irregular, to diverge downward, and probably to narrow downward into dikes. On the 400 to the 700-ft. level they are 300 to 500 ft. apart, and on the 500 and 700-

ft. levels are known to be connected by broad dike-like bodies and thus to surround a mass of breccia which underlies the Captain and Hidden Treasure claims.

Compression accompanied by a small amount of shearing developed a master fissure along the southward or southeastward trending part of the granite-breccia contact and a major tributary fissure of northward trend in the breccia closed the irregular edge of the eastern mass of latite-phonolite. Repeated disturbance took place along these fissures and was accompanied, in some places at least, by nearly horizontal slickensiding and slight shearing. Minor fissures directly or indirectly connected with both these fissures were formed during one or another of these disturbances. A comparatively early disturbance admitted phonolite dikes along both of the large fissures. Earlier syenitic and later basaltic dikes are found in the northward part of the master or contact fissure zone below the 1000-ft. level, and the basaltic dikes are also found at higher levels to the south. This distribution indicates a general northward pitch for the basaltic dikes as a whole, and their discontinuity implies that the disturbance which admitted them was a minor one.

Both of the major fissures and many of the minor fissures were traversed by vein-forming solutions and the thorough exploration of veins has thrown considerable light upon structural conditions. The main Portland vein fills the fissure along the granite-breccia contact from the 3000-ft. level up to the 600-ft. level, where the slope of the contact becomes less steep. There the vein passes upward into a group of veins, some in breccia and some in granite, and most of them with strikes nearly parallel to that of the contact, except where they approach the southward to eastward turn of the contact. There some of them cross the contact and continue along north-south fissures in the granite, and others branch off to the southeast and truncate the turn of the contact, pinching out within a very short distance after the eastward trending contact is reached.

The mining of these branch veins and of the veins in the granite shows that, regardless of the pitch of local ore shoots, the general pitch of the productive parts of the veins is northward and practically continuous with the main Portland vein, as shown in Fig. 8. The southern limit of stoped ground in the main vein closely parallels the southward to eastward bend of the granite-breccia contact for a considerable distance.

During 1926 prospecting southward was begun from the Portland 2100-ft. level along one of the veins in granite in Independence ground. The vein material is considerably oxidized and some encouraging assays have been obtained, but there has been no production unless very recently. It remains to be seen whether this vein is prevailingly lean or whether an interval of comparatively tight ground exists between the old workings above and productive ground below. This prospecting thus far has

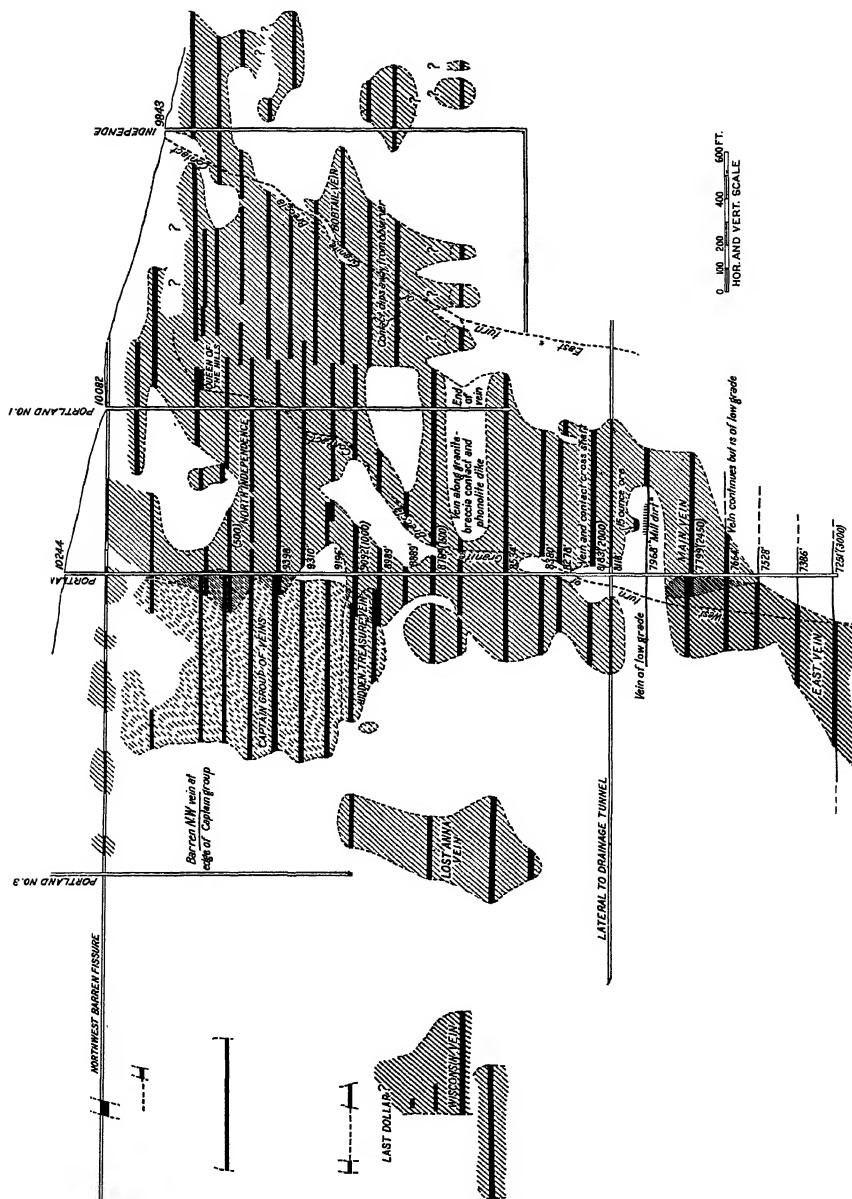


FIG. 8.—PROFILE OF PRINCIPAL VEINS IN PORTLAND AND INDEPENDENCE MINES.

Heavy solid lines represent stope lengths on different levels. The outline between levels is inferred and may include local barren spots.

been to the south of the granite-breccia contact. Prospecting toward the contact would be interesting as the vein in granite trends toward a branch of the main Portland vein which truncates the turn in the granite-breccia contact—a condition similar to that on the more productive levels of the Independence mine.

In the upper levels of the Portland mine, where the main vein is above the granite-breccia contact, the vein is not continuous northward nor northwestward beyond the No. 2 shaft, and has no apparent connection with the short veins of northwestward trend in the Captain and Hidden Treasure claims north and northwest of the shaft; but a connection becomes more apparent with increasing depth and is best shown on the 500 to 800-ft. levels. There the major tributary fissure sealed by a phonolite dike which approximately borders the eastern mass of latite-phonolite is not mineralized except for short distances where the short veins of northwest trend end against it, as shown in Fig. 9.

The distribution of tight and open fissures as indicated by pay shoots of ore is such as would result from a slight northward movement of the block on the east sides of the main or contact fissure and the major tributary fissure, the block squeezing tightly against the tributary fissure but leaving the main fissure open. At the same time the narrow mass of breccia between the two masses of latite-phonolite would become fractured obliquely by the shearing movement. The main vein northwest of the No. 2 shaft on these levels may be regarded as the westernmost of the group of short veins in the breccia which contained rich ore and were separated by mineralized ground of low grade which has at times been mined as mill ore. These veins continued productive down to the 900-ft. level and then pinched downward, as the latite masses narrowed and diverged and the local cause of fissuring died out. The major tributary fissure and its phonolite dike have not been found below the 900-ft. level, and the dike is evidently a branch of the phonolite dike which is exposed along the main or contact vein from the 800 down to the 3000-ft. level.

Below the range of this local condition the main vein still pinches out a short distance northwestward from the granite-breccia contact, but the principal Hidden Treasure vein, the Lost Anna vein and a few minor veins may be regarded as continuations of it in steplike arrangement. The two veins named cut the western mass of latite-phonolite and have been stoped more or less continuously from the 1000 to the 1800-ft. levels. The main vein has been stoped continuously below the 900-ft. level, but the ground between the 900 and 1200-ft. levels was so poorly mineralized in comparison to that above and below that it has been called "the barren zone."

From the 1800 to the 2000-ft. level stoping has been confined to the main vein, but on the 2000-ft. level the first indication is found of the "east vein" group, from which the principal production of the mine has

come during the last few years. This group trends a little east of north, and leaves the main vein a short distance north of the No. 2 shaft. One vein in this group was stoped for more than 100 ft. horizontally on the

load. BHP.



FIG. 9.—FIVE-HUNDRED-FOOT LEVEL OF PORTLAND MINE, SHOWING RELATION OF THE MAIN VEIN GROUP AND THE HIDDEN TREASURE AND CAPTAIN GROUPS TO A BARREN NORTH NORTHEAST FISSURE ALONG A PHONOLITE DIKE.⁹

2100-ft. level, but further development along that level proved it to be short and suggested that it and a few other minor veins represented the upward feathering out of the vein system. The "east vein" group is

⁹ After F. L. Ransome, U. S. Geol. Surv. *Prof. Paper* 54 (see Reference No. 1), and a more recent geologic map of the Portland Gold Mining Co.

not well exposed above the 2600-ft. level, but from there to the 3000-ft. level it consists of a continuous vein ("East No. 1") with a north pitch-

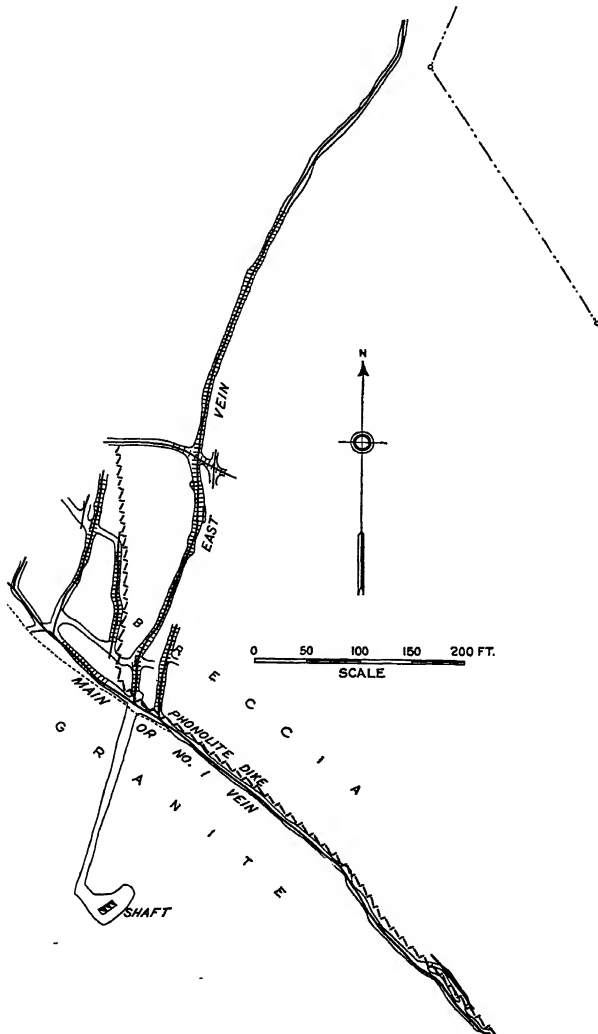


FIG. 10.—PLAN OF PORTLAND 3000-FT. LEVEL, SHOWING MAIN VEIN AND THE EAST VEIN GROUP.¹⁰

ing ore shoot, which has been followed on the 3000-ft. level for about 800 ft. from the main vein, and five or more rather closely-spaced parallel

¹⁰ From a map of the Portland Gold Mining Co., Nov., 1925.

veins which extend for short distances on the west (Fig. 10), and all of which branch from the main vein.

The productive part of the main vein, which has been stoped continuously with a northwestward pitch, does not extend appreciably beyond the east vein group, and on the 3000-ft. level its horizontal or strike length practically coincides with the width of the east vein group. The phonolite dike which has followed the main fissure so persistently leaves it and follows the East fissure zone. In other words, the direction of repeated disturbance leaves the granite-breccia contact and extends northward and downward along the East fissure.

Workings on the lowest levels of the Portland mine are not sufficiently extensive to permit a definite conclusion as to the local causes of fissuring. The granite-breccia contact where exposed on the 2450 to 3000-ft. levels has a northwest-southeast trend and there is some uncertainty on the 3000-ft. level as to whether this trend indicates an approach to the eastward turn that is exposed at higher levels near Independence ground (Fig. 2) or the westward turn toward Ajax ground. According to the first and less probable suggestion, further development should find that the contact turns from northwest to north or east of north and that the steep westward dip of the east vein will carry it downward to the contact.

In other words, the east vein should truncate the turn in the contact, duplicating the structure present on shallow levels near the Portland-Independence boundary, and should continue northward along the contact. According to the second suggestion, the east vein extends northward away from the granite and towards the syenite plug that crops out near the Last Dollar and Vindicator mines. This syenite has not yet been mapped on the lower levels of the Last Dollar mine or in the Roosevelt drainage tunnel (Portland 2100-ft. level), but its presence there and in the Wisconsin claim on the Portland 1700-ft. level indicates a downward broadening of the mass and a probable connection with the small syenite mass that crops out around the Rose Nicol mine. Compression induced by the syenite intrusion should produce transverse fissures about parallel to the east vein fissure in the downward-narrowing mass of breccia between the syenite and granite, and any subsequent compression or recoil due to disturbances of local or regional origin would keep those fissures open.

According to this interpretation the east vein should be continuous northward to and perhaps into the syenite. Its trend toward the Wisconsin or Modoc vein suggests a common local source for the two and encourages deep development of the Wisconsin vein in and near the Wisconsin and Last Dollar claims. The downward tapering of the breccia between syenite and granite implies a gradual shortening of the east vein within the breccia, but even so the cost of deep mining will continue for some time to be of more consequence than the persistence of

the vein. The east vein group may converge downward into one vein, and it remains to be seen whether the one vein can be profitably mined as distance from the contact increases. There is no indication that the character of the ore will change markedly with increasing depth. All veins must end downward eventually, and there must have been some depth below which tellurides could not have formed during the period of ore deposition; but these limits are indefinitely below the 3000-ft. level of the Portland mine.

Another question for speculation is whether or not the disturbance that formed the east vein formed parallel veins to the east and west of it, and if so whether or not any of them contain ore. No answer is justified until more is known of the boundaries of the granite and syenite.

DISTRIBUTION AND LOCAL SOURCES OF ORE IN THE VINDICATOR AND GOLDEN CYCLE MINES

The Vindicator No. 1 shaft is four-fifths of a mile northeast of the Portland No. 2 shaft. Its veins, the most persistent of which have been mined to a depth of 2000 ft. or more, are closely related to contacts between syenite masses and breccia.

Although syenite was mapped on the surface as a continuous mass of considerable size extending westward from the Vindicator No. 1 shaft, Lindgren and Ransome found that in the Vindicator workings it occurred as a group of small masses of irregular to dike-like form. This condition was confirmed by T. S. Lovering, who assisted the writer for a short time in 1925 and mapped the eighth, sixteenth, eighteenth, nineteenth and twentieth levels of the mine. Lovering found that the No. 1 shaft was in a local main plug of syenite of irregular shape, at least 600 ft. long and 300 ft. wide on the twentieth level and at least 800 ft. long and 500 ft. wide on the fourth level. Its longest dimension trends north 60° west with branches and small, apparently disconnected flanking masses trending either parallel to it or north 35° west. His maps of different levels, of which Fig. 11 is representative, suggest that the syenite was intruded along a broad fissure zone in breccia as a network of bulging dikes which converged locally to form the plug and which elsewhere partly or completely inclosed large slabs of breccia. Dikes of latite-phonolite, phonolite, and basaltic rock are present in their usual relationships.

The veins comprise three closely-related groups, northeastern, middle and southwestern. The northeastern group includes the Lillie and Nos. 1, 2 and 3 veins, which trend about north 60° west as a whole although the Lillie vein trends about north 45° west on the lowest levels. The middle group, locally called the "middle vein," includes on the lower levels a main vein trending north 30° west with short branches, mostly on its eastern side, trending south 60° east. On the eighth and fourth levels,

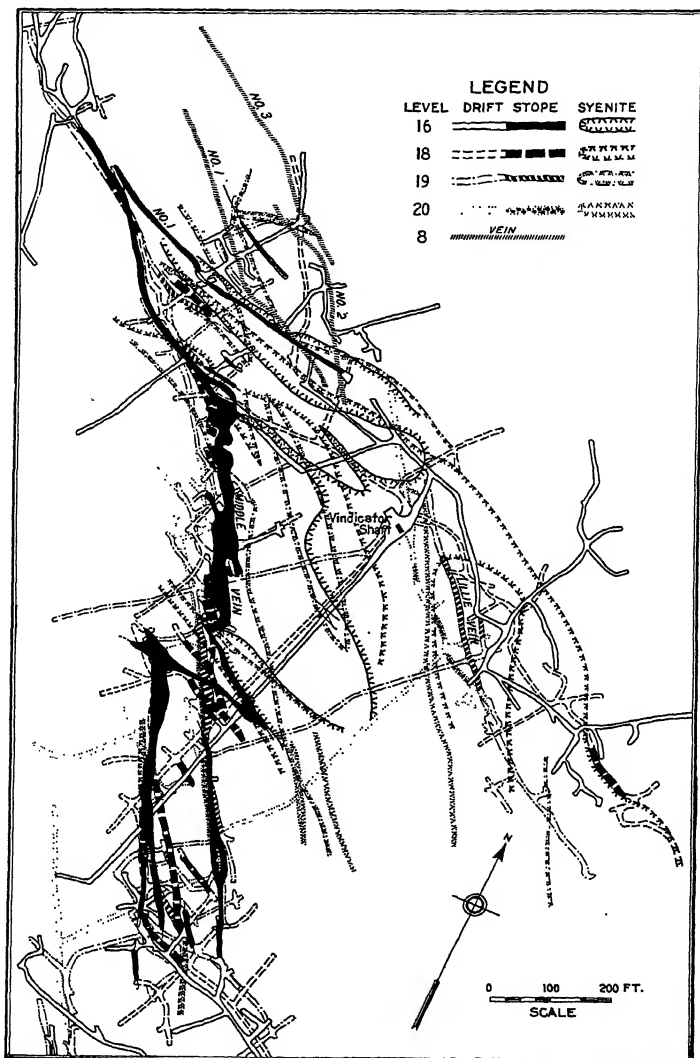


FIG. 11.—COMPOSITE PLAN OF VINDICATOR SIXTEENTH, EIGHTEENTH, NINETEENTH AND TWENTIETH LEVELS, SHOWING RELATION OF VEINS TO MARGINS OF SYENITE INTRUSION.¹¹

¹¹ Mapping by T. S. Lovering, supplemented by geologic data supplied by Al Beebe, superintendent.

the only upper levels accessible from the shaft, only short veins trending south 60° east are developed and form a steplike series which has a general trend parallel to the main vein below. The southwestern group is comparatively small. It parallels the "middle vein" and is connected with it by cross-veins that trend south 60° east.

The vein fissures are closely related to the main and minor syenite masses and here and there to dikes of other rocks, but the main syenite plug was the principal factor in determining the positions of the fissures. The northeastern group of veins trends west northwest closely parallel to and in part along the northeast contact of the syenite plug and a large outlier to the northwest. The group diverges northwestward, the No. 2 and No. 3 veins apparently dying out, and the No. 1 vein joining the middle vein in the northern part of Vindicator ground, and continuing with a north northwest trend. The No. 1, No. 2 and No. 3 veins dip southwestward, very steeply on the upper levels but less steeply on the lower levels. The Lillie and Lillie No. 2 veins, which represent a south-eastward continuation of the northeast group, are nearly vertical except on the nineteenth and twentieth levels where the dip is 65° southwest, or downward toward the middle vein; but this lowest part consists of narrow parallel veinlets of dense barren fluorspar and quartz. The ore shoots of the northeast group as a whole pitch northwestward, or toward the junction of the No. 1 and middle veins.

The main middle vein trends about north 30° west. It diagonally cuts a mass of breccia that lies between two prongs of the syenite plug on the 1600 and 1800-ft. levels, but nearly parallels the syenite contact on the 1900-ft. level. Its short branches parallel or coincide with the margins of the prongs above the 1900-ft. level, and there is some suggestion in the small amount of ground accessible that farther to the southeast the main vein closely parallels the margin of the syenite plug. The general pitch of ore shoots in the middle group is also steeply northwestward. The southwestern group is practically a local branch of the middle group and is governed by similar structural conditions.

The general northwestward pitch of the larger stopes in all the veins and the southeastward shallowing of workable parts of the veins indicate that vein-forming solutions ascended from the northwest, and that the most encouraging places for deeper prospecting are in that direction; but the distribution of syenite complicates the problem. Where veins in the mine have penetrated syenite they have in general become lean, and the poor results of development beneath the junction of the No. 1 and middle veins on the sixteenth level, at present inaccessible, may be due to this cause. The large stope of low-grade ore on the nineteenth and twentieth levels of the middle vein southwest of the shaft and the downward convergence of this part of the middle vein with the Lillie vein suggest that a main channel of circulation may have been directly beneath.

Only a small part of the middle vein was accessible on the twentieth level, the level of the drainage tunnel, and the only deeper work is a drift along the Lillie vein for 650 ft. on the twenty-first level, where the grade of ore did not encourage mining below drainage level.

There is no reason why the distribution of syenite and breccia at deeper levels should not be similar to that already exposed and that fissures favorable for ore shoots should continue to occur along the contacts. The syenite masses may coalesce downward for a distance and diverge again, producing a lean zone between the present workings and a lower zone of commercial grade. This question can be answered only by exploration. The arrangement of the veins indicates that the middle vein with its northward continuation towards Hull City ground is the main vein of all three groups, and any further deep exploration should be conducted along this vein either below the stope on the twentieth level or to the northwestward.

The Golden Cycle mine was largely inaccessible in 1925, but to judge from mine maps, its veins are an upward and southeastward continuation of the middle vein group of the Vindicator mine. Mining extends down to the Roosevelt drainage tunnel where the veins have evidently decreased considerably in number and extent. If the granite-breccia contact has about the same degree of slope in this vicinity as at the Independence and Portland mines it cannot be far from the bottom of the Golden Cycle shaft. This probability together with the evident connection of the Golden Cycle veins with the Vindicator veins and the prevailing northwestward pitch of the ore shoots offer little encouragement for deeper work in the Golden Cycle mine.

DISTRIBUTION AND LOCAL SOURCE OF ORES IN EAGLES MINE AND ADJACENT MINES

Recent deep exploration on the 1700, 1900, and 2100-ft. levels of the Eagles shaft of the Stratton estate has taken place along the Findley, Eagle, Rice, Pikes Peak and Porcupine veins and closely-related cross-veins, but without much success thus far. The vein fissures on these deep levels in part cut breccia and in part follow dikes of syenite, phonolite and in a few places latite-phonolite. The simple dikes of latite-phonolite on the low levels are in marked contrast to the large masses of this rock on the surface and at upper levels.

The intersecting systems of veins are shown in Fig. 12. The main veins for the most part trend north northwest and the cross-veins north northeast as far as the Isabella basaltic dike, which is close to the Eagles shaft on the upper levels and strikes east northeast; but to the north of this dike, the Zenobia vein, which is practically a continuation of the Pikes Peak vein, trends north northeast and, according to mine maps, is accompanied by short cross-veins of northeast trend.

The most extensive ore shoot thus far found on the low levels is at the 1700-ft. level in the Findley vein, which there coincides with a syenite dike and is according to Lindgren and Ransome's map,¹² a north-westward continuation of the Vindicator group of veins. The Findley vein has also been stoped on the 1500-ft. level, especially at its intersections with cross-veins, and is productive farther north at these deep levels

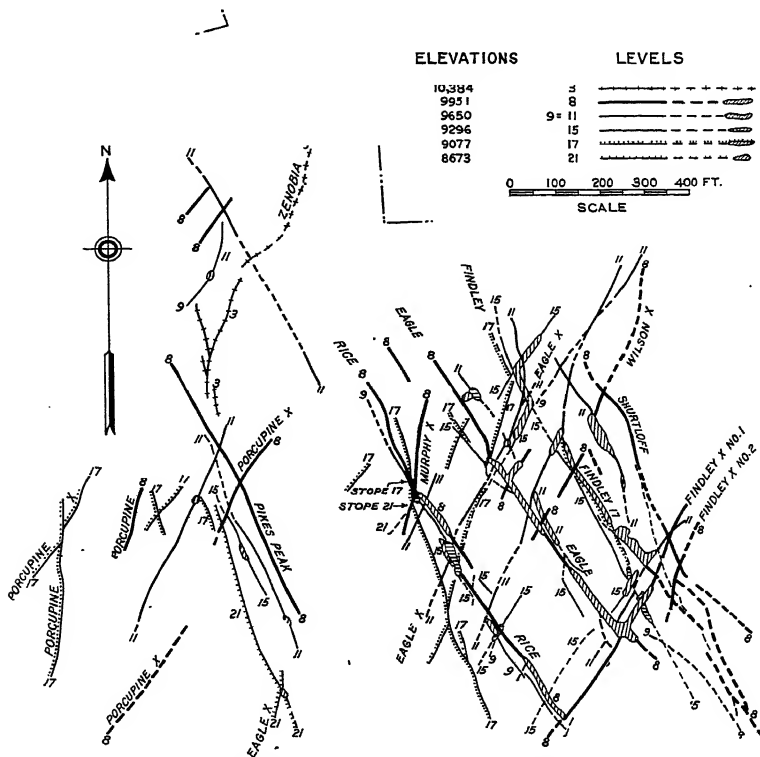


FIG. 12.—PLAN SHOWING DISTRIBUTION OF VEINS IN VICINITY OF THE EAGLES SHAFT.

than any of the other veins mentioned. The other veins, productive at the higher levels, are mostly tight at the low levels, but contain a few small ore shoots, some of high and some of low grade, at or near intersections with some of the cross-veins. The general pitch of the ore shoots, according to available data, is southward. The barren or very low-grade parts of these veins consist of parallel fissures 1 to 4 or 5 ft. apart connected by closely spaced oblique fractures indicative of horizontal shearing with little or no appreciable faulting.

¹² W. Lindgren and F. L. Ransome: *Op. cit.*, Plate 3.

The Eagle vein contains a "million dollar" shoot between the 800 and 1100-ft. levels. This shoot was continuous with a shoot in the Findley cross-vein which extended north northeast toward the main Findley vein. The Eagle vein has not been worth following any great distance southeast of the Findley cross-vein, and evidently derived its ore by upward circulation from the main Findley vein along the Findley and Wilson cross-veins, both of which are barren to the west of the Eagle vein.

The Rice vein is subject to a similar interpretation, though supported by less evidence. The larger stopes mined along it have been at or near intersections with cross-veins above the 1500-ft. level, and the southernmost stope, on the 800-ft. level, ends at the Findley cross-vein, which is there barren.

The Pikes Peak vein has been mainly productive above the 500-ft. level, though a few scattered small stopes have been found below, the lowest at the intersection with the Eagles cross-vein on the 2100-ft. level. The prevailing southward pitch of ore shoots in the mine and the finding of good ore in the Pikes Peak vein down to the sixth level of the Lucky Guss mine to the south,¹³ together with the comparatively small amount of development shown on available mine maps, suggests that more southward development at intersections with cross-veins in the upper and intermediate level is justified. Nowhere in the Eagles mine has the vein been followed south of its intersection with the Eagle cross-vein.

There is less evidence concerning the Pikes Peak vein than the veins to the east to permit a conclusion as to derivation of ore shoots, but such as exists favors the suggestion that solutions rose from the southward, and were conducted by cross-veins from trunk channels represented by the Last Dollar-Orpha May and Vindicator-Findley veins. The Pikes Peak vein, however, is so far from these trunk veins, and structural details between it and the Orpha May workings are so little known that this suggestion is not to be accepted too hastily. A cross-cut on level 13 from the Orpha May shaft¹⁴ crossed the position of the Pikes Peak vein near where its intersection with the Findley cross-vein would be expected, but evidently nothing encouraging was found. If, however, the Orpha May workings should be reopened this cross-cut will be a convenient starting point for the prospecting of the Pikes Peak vein at intersections with cross-veins.

The Porcupine vein, which is a northern continuation of the Orpha May vein, was not productive at the upper levels near the Eagles shaft according to available information,¹⁵ and a drift along it for 500 ft. on the 1700-ft. level found very little vein material. It was apparently

¹³ W. Lindgren and F. L. Ransome: *Op. cit.*, 383.

¹⁴ W. Lindgren and F. L. Ransome: *Op. cit.*, Plate 3.

¹⁵ W. Lindgren and F. L. Ransome: *Op. cit.*, 382.

productive on the sixth level of the Porcupine incline, 600 ft. south of the Eagles shaft, and the Orpha May vein was opened for 1000 ft. farther south in the Orpha May mine, where it was productive on the upper levels, especially at intersections with cross-fissures. The vein was there developed to a depth of 1260 ft., but little ore was reported from the lower levels.

Although this vein has been called a trunk vein, the localization of ore along it to intersections with cross-fissures implies that it was kept rather tight by the compression or shearing that preceded ore deposition. It does not imply, however, that ore shoots did not form well below the deepest workings, and it may be reasonably inferred that the ore was derived from the same local source as the ore in the Portland mine, deep below the Wisconsin and Last Dollar claims; but the relatively small ore shoots thus far found also justify the inference that a smaller supply of ore-forming solution took this northward course, and that less favorable structural conditions caused it to disperse more widely than in the Portland mine.

The stage of fissuring that preceded ore deposition in the vicinity of the Eagles mine was evidently due to compression and local shearing of the breccia between granite on the north and the syenite mass between the Last Dollar and Vindicator mines on the south. Major breaks took place along the Last Dollar-Orpha May and the Vindicator-Findley fissure zones at each end of the syenite. The syenite and the breccia immediately north of it, despite the undoubted existence of many local fissures, acted essentially as one block which was bounded on the north by a cross-fissure parallel to or coincident in strike with the Findley cross-vein. Compression kept this cross-fissure generally tight, and it was therefore unfavorable for the formation of ore shoots except near its junction with the main Findley vein, where compression was locally relieved. The ground to the northwest of this cross-fissure was broken into a group of diamond-shaped blocks bounded by the fissures of the Eagles mine. Shearing along the cross-fissure tended to open the adjoining parts of the north northwest fissures, and local east-west stretching complementary to the north-south compression tended to open the north northeast fissures near the main Findley vein. These tendencies increased as weight of overlying rock decreased. Ore shoots of considerable size could thus form farther and farther northward from the cross-fissure and westward from the main Findley vein as ore-forming solutions rose toward the surface, and tended to localize at and near the intersections of fissures.

According to this interpretation of veins near the Eagles shaft, deep prospecting is of more promise southward toward the Findley cross-vein than in the opposite direction. Deeper prospecting than heretofore along the Orpha May vein is worthy of consideration especially in the

direction of the Last Dollar mine. The main Findley vein is especially worthy of attention at deeper levels than it has been worked thus far, and the part of it extending southward from a point due east of the Eagles shaft to Vindicator ground is one of the more promising places for deep mining. The local deep source of the ore in this and the related veins in Eagles ground is, to judge from the general downward convergence of ore shoots, near the boundary between Vindicator and Hull City ground.

SUMMARY OF CONCLUSIONS REGARDING ORE AT DEEP LEVELS

Deep developments in the southeast quarter of the district indicate a few local sources from which ore-forming solutions rose and spread, and toward which ore shoots tend as a whole to converge downward. One of these sources, below the collapse breccia in the Dante mine, supplied the orebodies of the Cresson mine and the neighboring veins to the east. Development on the 1900 and 2000-ft. levels of the Cresson mine has been retarded by excessive amounts of water, but ore is expected to continue to considerably greater depth although its areal extent is expected to shrink toward the Dante claim. The neighboring veins are not so deeply developed as the Cresson orebodies and are worthy of deeper prospecting especially in the vicinity of the Dante claim. The southward continuation of these veins may have been fed from other local sources, which also supplied ore to the Ajax and neighboring groups of veins; but this part of the area has not been sufficiently studied to warrant a more explicit statement.

A second local source, which supplied ore to the Independence, Portland, Last Dollar, Orpha May and neighboring mines, is approximately below the Last Dollar mine. This source may also have contributed to the veins near the Eagles shaft. Ore shoots in this great system owe their shapes and distribution to strictly local conditions, but in general distribution they converge downward toward the source. There is still considerable promising ground above as well as below the drainage tunnel in the vicinity of the Wisconsin, Last Dollar, and Orpha May claims.

A third local source, which supplied ore to the Golden Cycle, Vindicator, Findley, Eagles and neighboring mines, lies approximately beneath the north boundary of Vindicator ground. Considerable ground above drainage level in this vein system remains to be prospected, and on the results to be obtained will depend the advisability of deeper prospecting.

The area studied includes the mines that have contributed the greatest part of the district's production. This statement implies that deeper development has correspondingly more promise in this part of the district than elsewhere, but should not discourage the consideration of deep

prospecting in the remainder of the district. No further comment is justified until adequate study of the rest of the district has been made.

The writer thus far has had little opportunity to study the grades of ore mined at different levels in the different vein systems, but so far as mineral composition of the ore is concerned there is no noteworthy change down to the deepest levels reached. The preceding paragraphs show that ore in certain restricted areas presumably of similar grade to that above is to be expected well below the present deepest levels, and the ultimate depth of mining in these restricted areas will be determined by the grade of ore and the cost of deep mining.

Acceleration Stresses in Wire Hoisting-ropes

BY G. P. BOOMSLITER,* MORGANTOWN, W. VA.

(New York Meeting, February, 1927)

IN previous discussions on stresses in hoisting ropes, little has been said concerning the effect of the elasticity of the rope itself on the stresses due to acceleration. Laschinger¹ has calculated acceleration stresses for several cases but he has ignored the elasticity of the rope as a factor, except as it compresses the drum and stretches itself as it winds thereon. He also deals with steam plants and he shows the effect of the "jerky" start of the engine. This effect would actually be reduced by the elasticity of the cable.

With the advent of the motor-driven hoist, new conditions have arisen. The jerk mentioned by Laschinger has been eliminated and smoother starting made possible. But the elastic properties of the rope seem to introduce considerable periodic acceleration stresses in the rope. The following discussion of acceleration stresses is based on the assumption of a constant acceleration at the drum. To be exact, the mathematical work is based upon the assumption that the top is moving upward with a constant acceleration, and ignores the fact that the cable is shortening as it winds. However, the effect of the shortening would be inconsiderable during the first part of the acceleration period.

Most hoisting engineers make the assumption of constant acceleration at the drum. Under the older method of calculating acceleration stresses, this would be true with a constant torque motor. But if stresses vary, as this paper attempts to show that they do, constant torque would not give constant acceleration, although the velocity-time curves obtained in the full size test herein described showed that they approach rather closely to this condition, at least as nearly as could be measured.

THEORETICAL DISCUSSION

According to elementary physics, the force required to accelerate a body of weight W at the rate of f ft. per sec.² is $\frac{Wf}{g}$. To raise the body with acceleration of f ft. per sec.² the total force required is $W + \frac{Wf}{g}$, in

* Department of Mechanics, West Virginia University.

¹ E. J. Laschinger: Discussion of Winding Plants for Great Depths by H. C. Behr. *Trans. Inst. Min. & Met.*, London (1901-2) 11, 272.

which $g = 32.2$ ft. per sec. per sec. Thus a force of about $\frac{5W}{4}$ is needed to raise a body with an acceleration of 8 ft. per sec. per sec.; similarly, if it were lowered, the force acting upon it when accelerating downward at 8 ft. per sec. per sec. would be about $\frac{3W}{4}$.

When engineers are making hoisting calculations for a mine hoist, they commonly assume a constant acceleration at the hoisting drum up to maximum speed and constant retardation from this speed to a stop. If the rope were absolutely stretchless, the cage would have the same acceleration as the rope which is being wound on the drum. But ropes are elastic. It is well known that any bar, or piece of material, under tensile stresses elongates. Consequently, there can be no stress without stretch and vice versa.

Suppose then that there be a constant acceleration at the drum and that the cage rests on the bottom of the shaft. The first effect of the winding of the drum is to take up the rope stretch so that the tension in the rope will first hold up the cage. As soon as more rope is wound up, more than enough stretch to give W lb. of pull is developed and the cage begins to rise with an acceleration beginning at zero and gradually increasing until it is the same as that of the drum. But meantime the drum has been speeding up at a constant rate, so that at the instant when the two accelerations are alike the rope at the drum is moving faster: *i. e.*, has a higher velocity than the cage. This stretches the rope still more, so that the cage acceleration becomes greater than that of the rope which is being wound up. Because of this higher acceleration the speed of the cage increases until it finally becomes that of the drum. However, by this time the rope at the drum has moved much farther than has the cage. Consequently there is more stretch in the rope than is needed to give the cage the acceleration of the drum and it soon moves faster than the drum, thus reducing the stretch and the acceleration at the cage, so that ultimately its acceleration and speed are again reduced below those of the drum, and the cycle begins again.

Acceleration-time Curves

This cycle may be clearer by studying the accompanying acceleration-time, velocity-time and space-time curves for such a motion (Fig. 1), in which it is assumed that the cage hangs freely near the bottom of the shaft. The lower curves represent the acceleration-time curves at the drum and cage respectively, the straight line represents the constant acceleration of the drum, and the curve, the variable acceleration of the cage. After t_1 seconds the accelerations of the two are equal, represented by the ordinate AB .

Mechanics proves that the area under the acceleration-time curve during any period represents the change in speed during that period. As both began with zero velocity, the area under the curve for the time t_1 gives actual velocities. Obviously the area $ABCD$ under the straight line is larger than the area ABC under the curve. Hence the velocity of

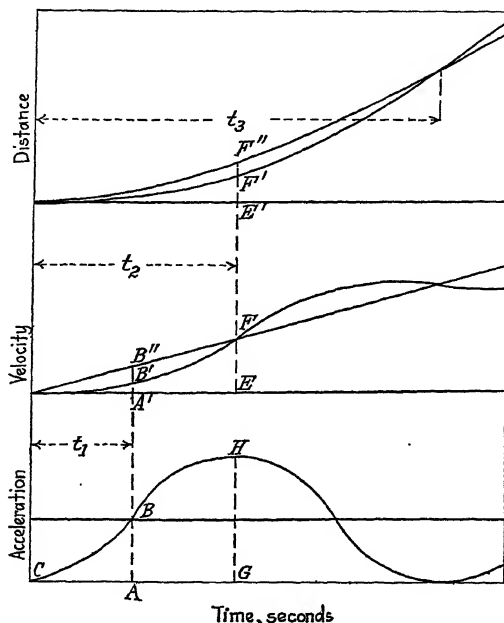


FIG. 1.—ACCELERATION-TIME, VELOCITY-TIME AND SPACE-TIME CURVES SHOWING HARMONIC VIBRATION OF A CAGE HOISTED BY AN ELASTIC CABLE.

the drum must be greater than that of the cage and the cable must be stretching, thus increasing the pull in it and thereby the acceleration of the cage.

Velocity-time Curves

The second set of curves are the velocity-time curves; the straight line for the drum and the curve for the cage are derived from the areas under the previous set of curves. According to the conclusion there reached $A'B'$, which represents the velocity of the cage, is less than $A'B''$, which represents the velocity of the drum after t_1 seconds.

Distance-time Curves

Because of the increased stretch and the consequent increased acceleration of the cage, it finally reaches the velocity of the drum after t_2

seconds. This common velocity of drum and cage is represented by the ordinate EF . But until this time the velocity of the cage has been continually less than that of the drum. Furthermore, the area under the velocity-time curve gives the distance moved. Hence the drum has moved farther than the cage, thus continually stretching the rope, as is shown graphically in the upper sketches, which are the distance-time graphs for the motion. $E'F''$ represents the distance moved by the drum, $E'F'$ the distance moved by the cage and consequently $F'F''$ represents the stretch. But at this instant the velocities are equal and the acceleration of the cage much greater than that of the drum. Hence it will catch up on the drum, reducing the stretch and acceleration, until this falls first to that of the drum and finally to zero and the cycle begins again.

The greatest acceleration of the cage is represented by the ordinate GH in the lower graph and is of course proportional to the stretch $F'F''$ in the upper graph. It will be noted that the motion resembles a simple harmonic motion and mathematical analysis shows that it is one. Now stress is proportional to stretch, so the stress will vary and will be greatest when the stretch is greatest.

If the weight of the cable is neglected, the mathematically derived expression for the greatest stress set up in a cable when the top is accelerated with a constant acceleration is:

$$S_{max} = W \left[1 + \frac{f}{g} + \frac{\sqrt{f^2 + 2fg(m+1)}}{g} \right] \quad (1)$$

in which W is the weight of the cage, f ft. per sec. per sec. is the constant acceleration of the top, g is the acceleration due to gravity = 32.2 ft. per sec. per sec., and m is the ratio of the slack or stretch in the cable to its stretch under the weight of the cage. If we assume no stretch or slack then

$$S_{max} = W \left[1 + \frac{f}{g} + \frac{\sqrt{f^2 + 2fg}}{g} \right] \quad (2)$$

If the cage hangs freely suspended equation (1) shows that the stress due to acceleration is twice that found by the commonly accepted formula $F = ma$.

The derivations of these formulae will be found in the appendix to this paper.

Fig. 2 shows graphically how this acceleration stress increases as the acceleration increases for the case in which the cage rests on the bottom with no stretch in the cable. It is a graphical representation of equation (2). Note that at an acceleration of 8 ft. per sec. per sec., the greatest stress induced in the cable = $2W$ instead of $1\frac{1}{4}W$ as it would be by the common theory. For an acceleration of 16 ft. per sec. per sec.,

such as is used in some of the coal mines in this country, $S_{max} = 2.6W$ instead of $1\frac{1}{2}W$ by the old $F = Ma$ formula.

If in the mathematical discussion, we consider slack or stretch in the cable and also its weight, the expressions become complicated. Table 1 shows a series of data from such a discussion for various accelerations

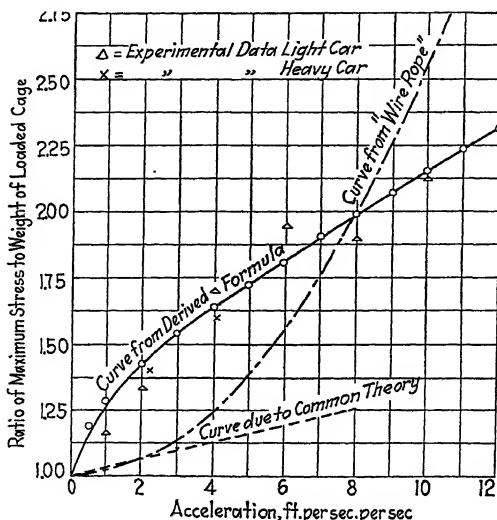


FIG. 2.—ACCELERATION STRESSES ON A LOADED CABLE:—, Curve from derived formula; ———, curve from "wire rope;" ———, curve due to common theory; Δ experimental data, light car; X, experimental data, heavy car.

and weights of cable with varying values of slack or stretch. A study of the resulting stresses in the rope will be illuminating. For instance the total stress in a rope that weighs $\frac{1}{6}$ as much as the cage ($n = 6$), at an acceleration rate of 8 ft. per sec. per sec., will be $2.28W$ instead of $1.46W$ by the common theory. If the rope is slack when the cage is lifted, these stresses will be greatly increased.

EXPERIMENTAL EVIDENCE

To verify the conclusions given, a series of experiments were made varying in complexity.

The first two were laboratory experiments, in which a relatively heavy car on an inclined plane was used to obtain a constant acceleration. A cord was led from the car over a pulley to the elastic member which was a piece of rubber hose in one instance and a helical steel spring in the other. A weight much smaller than that of the car was suspended

from the lower end of the elastic cable. The car was allowed to run down the plane and the greatest stretch in the rubber hose was measured during the run. The ratio of this stretch to the stretch caused by the static suspended weight gave the ratio of the maximum tension during acceleration to the dead load tension.

TABLE 1.—*Ratios of Maximum Cable Stresses to Load W for Various Accelerations, Stretches or Slacks and Cable Weights*

Values of n	$M = +1$ Acceleration in Ft. per Sec. ²						Values of n	$M = +\frac{1}{2}$ Acceleration in Ft. per Sec. ²					
	2	4	6	8	10	12		2	4	6	8	10	12
1	2.89	3.36	3.69	4.05	4.38	4.80	1	2.81	3.22	3.55	3.90	4.21	4.54
2	2.20	2.56	2.87	3.14	3.43	3.64	2	2.14	2.46	2.74	3.02	3.30	3.48
3	1.99	2.33	2.61	2.83	3.10	3.33	3	1.92	2.23	2.49	2.73	2.97	3.17
4	1.90	2.22	2.49	2.73	2.95	3.17	4	1.83	2.12	2.37	2.57	2.81	3.03
5	1.81	2.13	2.40	2.67	2.85	3.06	5	1.75	2.04	2.29	2.50	2.72	2.90
6	1.76	2.09	2.34	2.60	2.79	2.94	6	1.67	1.99	2.22	2.46	2.67	2.82
8	1.72	2.04	2.27	2.50	2.71	2.91	8	1.66	1.94	2.15	2.41	2.57	2.75
10	1.72	2.03	2.22	2.46	2.67	2.85	10	1.57	1.90	2.10	2.30	2.52	2.69

Values of n	$M = 0$ Acceleration in Ft. per Sec. ²						Values of n	$M = -1$ Acceleration in Ft. per Sec. ²					
	2	4	6	8	10	12		2	4	6	8	10	12
1	2.72	3.10	3.41	3.74	4.01	4.32	1	2.50	2.77	3.03	3.30	3.54	3.81
2	2.05	2.32	2.58	2.86	3.08	3.30	2	1.80	2.02	2.21	2.38	2.58	2.77
3	1.84	2.10	2.36	2.56	2.77	2.97	3	1.59	1.76	1.94	2.11	2.25	2.46
4	1.75	2.01	2.24	2.38	2.63	2.76	4	1.48	1.65	1.81	1.96	2.11	2.27
5	1.67	1.93	2.15	2.35	2.53	2.71	5	1.41	1.57	1.73	1.88	2.02	2.16
6	1.62	1.86	2.08	2.28	2.46	2.64	6	1.36	1.51	1.66	1.82	1.95	2.09
8	1.58	1.81	2.02	2.22	2.40	2.56	8	1.31	1.45	1.59	1.73	1.88	2.00
10	1.54	1.79	1.98	2.17	2.35	2.50	10	1.27	1.40	1.54	1.67	1.82	1.96

M is the ratio of slack or stretch to elongation under dead load. This = $\frac{Wl}{aE}$.

n is the ratio of the load raised to the weight of cable. Thus, when $n = 8$, the cable weight is $\frac{1}{8}$ of the load.

The acceleration was found by measuring the time interval during which the car moved a known distance. Assuming constant acceleration, the magnitude of the acceleration was found by the well known formula $S = \frac{1}{2}(f)(t)^2$, in which S is the distance moved through in feet, f , the acceleration in feet per second per second and t the elapsed time in seconds. The acceleration could be varied by varying the slope of the inclined plane. The points represented by triangles in Fig. 2 show the results of this experiment.

In the second experiment, a longer inclined plane was used, and a heavier car. The cord, a wire in this case, passed over a pulley and down

from the roof of a three-story building to the sidewalk below where it was attached to a rubber hose as the elastic member, on the end of which was hung the weight. Instead of measuring stretch, however, a spring scale was inserted between the wire and the top of the elastic hose. When the car was allowed to run down the plane, the maximum scale reading was obtained. The results of this experiment are indicated by the crosses in Fig. 2. Unfortunately, it was impossible to attain very high accelerations because of the great momentum of the car at the end of the run.

FIELD TEST ON A MINE HOISTING-CABLE

Although the results of the experiments just described agreed closely with the mathematical theory, it was felt that tests on a rope under working conditions were desirable. Obviously, the only kind of apparatus that could be used was something that would give an automatic record. Furthermore, because of the rapid vibration there must be little inertia of the vibrating parts, and the apparatus must be accurate and sensitive.

Telemeter

The apparatus decided on was the telemeter, designed by McCollum and Peters at the Bureau of Standards. It uses the principle that the electrical resistance of a stack of carbon plates varies inversely as the pressure on the plates. It consists of a strain gage, or several of them, with a Wheatstone bridge for each with proper resistances and an oscillograph element to which is attached a camera equipped with a roll of film or sensitive paper upon which is photographed a beam of light reflected from a mirror. The vibration of the mirror deflects the beam of light and the deflection is proportional to the stress in the member tested. The strain gage is a U-shaped piece of steel, wrought from the solid, in which the carbon element is inserted. This gage is then attached to the bar, the stress in which is to be measured.

Fig. 3 shows a set of three strain gages attached to a mine hoisting-rope. Here the gage length was 4 in. although the instrument was designed for a gage length of 8 in. The instrument has been described by McCollum and Peters,² with a theoretical discussion of its working and suggested uses.

Mine Conditions Where Tests Were Made

Permission was obtained to use the shaft of the Ida May mine of the Consolidation Coal Co., on Helen's Run, about 8 m. west of Fairmont,

² Burton McCollum and O. S. Peters: A New Electrical Telemeter. Bureau of Standards, *Tech. Paper* 247 (1924) 40 pp.

W. Va. The mine was in working order but was shut down except for infrequent hoistings of coal and, therefore, lent itself admirably to the project.

The shaft is 550 ft. deep from sump rail to dump, or 507 ft. from bottom of coal to top of casing. The wire cable is of $1\frac{1}{2}$ in. diameter. The power for hoisting is obtained from a 1000-kw. direct-current motor generator set, direct connected to the hoisting drum and to an Ilgner

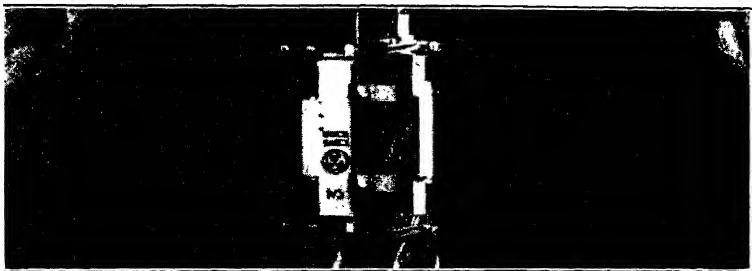


FIG. 3.—TELEMETER GAGES ATTACHED TO MINE CABLE.

flywheel. It is driven by a 600-hp. alternating-current motor. The hoisting engineer regulates the speed by a lever on the controller. The hoisting drum consists of one cylinder of 11 ft. diameter, another of 8 ft. diameter, and a conical drum between them on which the rope travels from one to the other in five turns. The hoist is double-compartment, the empty cage going down in one compartment while a load of coal is being hoisted in the other. The empty cage weighs 11,300 lb., the empty car 3500 lb. and the load of coal 6800 pounds.

Method of Tests

To attach the strain gages to the rope, 2 rings were clamped to it 4 in. apart. The gages were then clamped to rods extending from the rings in the direction of the axis of the cable. Three gages were used, at angles of 120° with each other, and were attached to 3 elements of the 6-element telemeter used. Fig. 3 shows the gages attached to the rope. The telemeter was placed in the cage or car, the camera motor started and the test was ready to begin.

Tests were made under 3 loads; the empty cage, cage and empty car, and cage and loaded car. Four runs up and down were made with each load. Fig. 4 shows the telemeter apparatus mounted on a car of coal.

The strain gage, of course, measures only strain, and the stress could not be found until the strain due to a measured load was determined. This was done by lowering the cage on landers and slacking off the rope

until there was no stress in it. The cage was then raised off the landers. The deformation due to this operation was shown in the recorded deflection

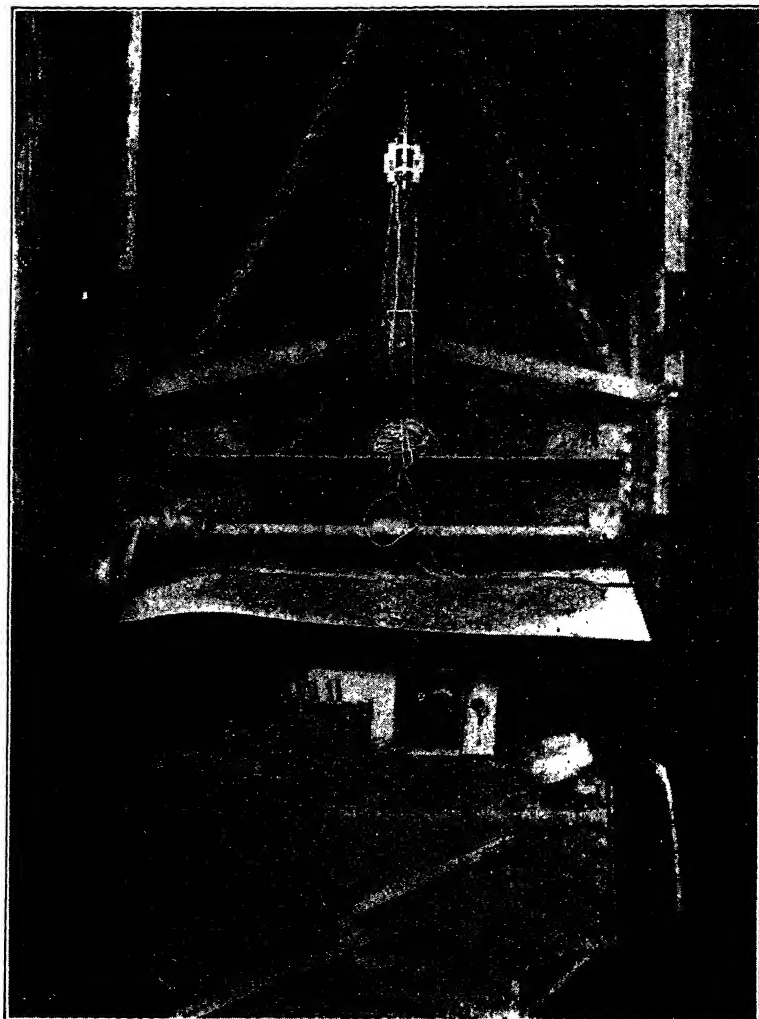


FIG. 4.—TELEMETER APPARATUS IN POSITION FOR TEST.

of the beam of light. The cage was then successively lowered and raised. Any excess deformation, then, over that obtained by raising the cage off the landers, was due to acceleration stress. According to theory there

should be a series of vibrations during acceleration. Figs. 5 and 6 are typical records showing the recorded deformations.

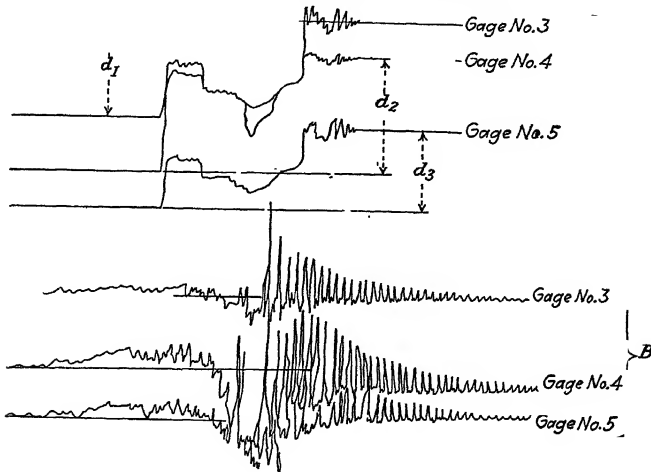


FIG. 5.—A, DEFLECTIONS WHEN CAGE AND LOADED CAR ARE RAISED FROM LANDERS; B, DEFLECTIONS WHEN CAGE ALONE WAS LIFTED AND CIRCUIT BREAKER WENT OUT.

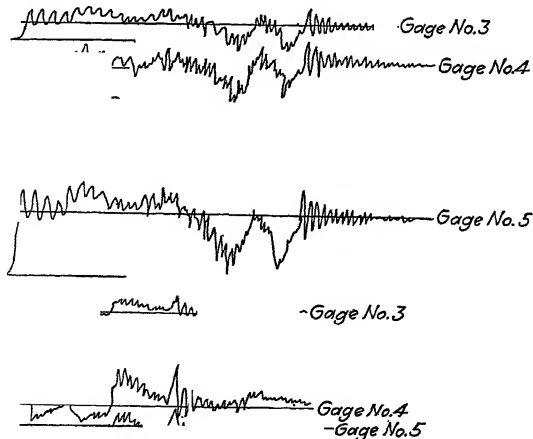


FIG. 6.—A, DEFLECTIONS SHOWING VIBRATION AT START WHEN HOISTING BEGINS FROM A STATE OF PARTIAL RELIEF OF LOAD; B, DEFLECTIONS SHOWING EFFECT OF SUDDEN BREAKING.

Discussion of Results

Stresses in pounds per square inch could be found by another method. The deformation scales for the three elements were not alike, neither were the deformations themselves when compared to scale. Therefore the average deformation for the three was obtained not only for dead load

but also for live load at various intervals. Knowing the load lifted and size of rope, the unit stress S due to the load is equal to the load divided by the cross-sectional area of the rope. Knowing the deformation in 4 in. the unit deformation e is obtained. Now $E = \frac{S}{e}$, in which E is the modulus of elasticity of the wire rope.

TABLE 2.—*Maximum Recorded Stresses in Test of Ida May Mine Hoisting-rope*

Trip No.	Average Deform. (in 4 in.), in.	Average Stress Increase, Lb. per Sq. in.	Maximum Observed Acceler- ation, Ft. per Sec. ²	Ratio of Maximum Stress to Weight			Remarks
				Observed	Formula 1	Common Theory	
Series 1. Empty Cage Only. Dead Load Unit-stress = 12,000 Lb. per Sq. In.							
Trip 1 down	0.000706	3,530		1.29			Decelerating
1 up	0.000885	4,425		1.37			Starting
2 down	0.00104	5,200	11.25	1.43	1.70	1.35	Decelerating
2 up	0.00309	15,450	4.25	2.29	1.27	1.13	C. B. went out
3 down	0.00132	6,610	6.25	1.55	1.39	1.19	Sudden stop
3 up	0.00112	5,600	5.00	1.47	1.31	1.16	Accelerating
4 down	0.00066	3,295	7.00	1.27	1.44	1.22	Stopping
4 up	0.000855	4,275	5.00	1.36	1.34	1.17	Accelerating
Series 2. Cage and Empty Car. Dead Load Stress = 15,600 Lb. per Sq. In.							
Trip 1 down	0.000857	4,285	7.00	1.27	1.44	1.22	Stopping
1 up							Data unreliable
2 down	0.00101	5,030	6.6	1.32	1.41	1.21	Decelerating
2 up	0.00146	7,300	9.5	1.47	1.59	1.30	Stopping
3 down	0.00085	4,240	5.5	1.27	1.34	1.17	Decelerating
3 up	0.00094	4,675	7.25	1.30	1.45	1.23	Stopping
4 down	0.00143	7,165		1.46			Decelerating
4 up	0.00123	6,440		1.41			Accelerating
Series 3. Cage and Loaded Car. Dead Load Stress = 22,900 Lb. per Sq. In.							
Trip 1 down	0.00114	5,685	2.75	1.25	1.17	1.09	Decelerating
1 up	0.0012	5,985	5.00	1.26	1.31	1.16	Accelerating
2 down	0.00146	7,300	7.00	1.32	1.43	1.22	Stopping
2 up	0.00117	5,870	4.00	1.26	1.25	1.13	Accelerating
3 down	0.00159	7,940	6.75	1.35	1.42	1.21	Decelerating
3 up	0.00123	6,160	6.50	1.27	1.40	1.20	Accelerating
4 down	0.0015	7,455	5.5	1.33	1.34	1.17	Decelerating
4 up	0.00107	5,360	5.5	1.23	1.34	1.17	Accelerating

Unfortunately the stress in the cable was not wholly relieved when the cage alone was on the landers, and this value of E was not obtained. In the other two cases, values of E of 18,750,000 lb. and 20,000,000 lb. per sq. in. were obtained. By the reverse process, measuring the increased deformation under motion, dividing by the gage length of 4 in. and multiplying by E would give the unit stress due to acceleration. Table 2 shows the results of these tests and compares them with that obtained from the formula $F = mf$ and also with formula (1) on page 77.

The accelerations at the drum were obtained by use of an electric tachometer. As this was not a recording instrument, readings were

recorded by an observer, at intervals of 1 sec. during each run. As these were in revolutions per minute, they had to be translated into feet per second. It was therefore necessary to know when the rope was on the 8-ft. drum, when on the 11-ft. drum and when on the cone. An observer called out when the rope passed from one to another and the record was taken. From this data, a velocity-time curve was drawn, the slope of which at any instant gave the acceleration at that instant.

To those familiar with the properties of hoisting ropes, a value of E of 20,000,000 lb. per sq. in. seems high. However, tests made by the writer (page 19) of five different sizes of wire hoisting-rope 5 ft. long on an 8-in.

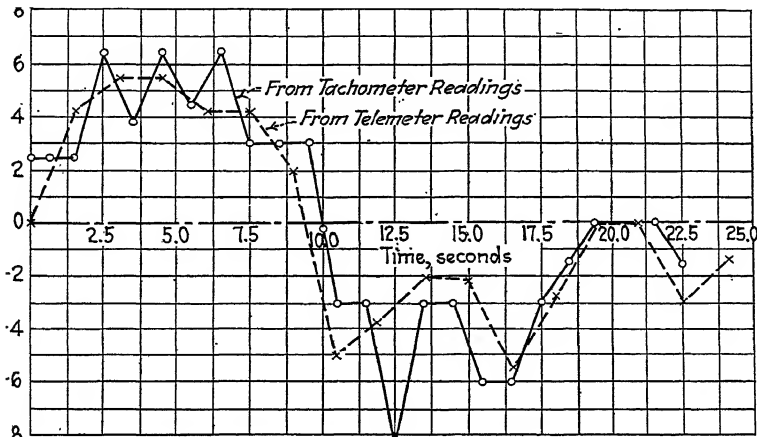


FIG. 7.—FOURTH RUN, SERIES 3, CAGE AND LOADED CAR. COMPARISON OF ACCELERATIONS AS DETERMINED BY TELEMETER AND TACHOMETER READINGS; CURVE A FROM TACHOMETER READINGS; CURVE B, FROM TELEMETER READINGS.

gage length gave values as high as 18,000,000 lb. per sq. in., after having been stressed beyond the elastic limit, and averaged about 16,000,000 lb. per sq. in.

It seemed advisable to check these values by comparing the acceleration as obtained from the tachometer readings with the average found with the telemeter. This was done as follows: enough accelerations were obtained from the velocity-time curve, derived from the tachometer readings, from which to draw an acceleration-time curve. A similar curve was drawn by observing average deformations at frequent intervals on the corresponding telemeter record and translating them into accelerations using the derived values of the modulus of elasticity. The two were then compared. Fig. 7 shows the two curves for an upward hoist with cage and empty car. The values of the acceleration and the forms of the graphs are remarkably similar.

Two incidental phenomena in this test are of interest. Fig. 5B is a record taken when the circuit breaker went out while the cage was moving upward, thus setting the safety brakes and suddenly stopping the car. The greatest stress recorded is 2.28 times the dead weight of the load hoisted. Fig. 6B shows the effect of suddenly applying the brakes. The greatest stress recorded was 1.55 times the weight of the suspended load. They would seem to indicate that the skill of the hoist operator might have considerable influence on the life of a hoisting cable.

CONCLUSIONS

Recent studies of repeated stress phenomena³ show that steel will withstand more than 100,000,000 repetitions of stress within the elastic limit without failure, but a much smaller number when the stress exceeds the elastic limit. This number rapidly becomes less as the intensity of stress increases. Hoisting ropes in constant service often last only 6 mo. The question may well be asked whether this short life is not due to excessive stress and this excessive stress itself due to a neglect of the full effect of acceleration stresses. Mine laws in many states and general practice recommend the use of a factor of safety of five. Calculations involving the theoretical acceleration stresses will reduce this in many cases until the factor is little more than two. The endurance limit of steel is little more than $\frac{1}{2}$ of the ultimate strength. Thus the greatest stress set up under normal conditions of hoisting is barely below the endurance limit in many cases.

For example let us consider the rope and hoist at the test mine. This was undoubtedly well designed by ordinary methods.

The load was 21,600 lb.; 559 ft. of cable weighs about 2000 lb. Then $n = \frac{21,600}{2000}$ say 10. Also one cage rests on the bottom when the other is in the dumping horns, but let us assume $m = +\frac{1}{2}$. The acceleration under working conditions is about 8 ft./sec². In the second series of Table 1, with $f = 8$ ft./sec², $m = +\frac{1}{2}$ and $n = 10$, we find $S_{max} = 2.30W$. Then the greatest stress due to weight and acceleration = $2.30 \times 21,600 =$

49,700 lb.

Add friction, say about

500 lb.

Add bending stress of $1\frac{1}{2}$ -in. cable on 8-ft. pulley = 9,440

Total

59,640 lb.

Proof load of $1\frac{1}{2}$ in. 6×19 plow-steel hoisting rope = 164,000 lb.

³ University of Illinois Experiment Station Bulletins: H. F. Moore and J. B. Kommers: An Investigation of the Fatigue of Metals. No. 124 (1921). H. F. Moore and T. M. Jasper: An Investigation of the Fatigue of Metals. No. 136 (1923), No. 142 (1924) and No. 152 (1925). H. F. Moore: Tests of the Fatigue Strength of Cast Steel. No. 156 (1926).

Therefore the safety factor is

$$164,000 \div 59,640 = 2.75$$

Thus, the original safety factor is cut nearly in half and any unusual load, acceleration, shock or starting speed may increase this to or above the endurance limit.

RECOMMENDED METHODS OF DESIGN OF HOISTING ROPE

As was pointed out previously, the formula for acceleration stress when the rope weight is considered is very complicated and therefore difficult for use in design. However, a simple empirical formula may be obtained. The ratio of the greatest stress to dead weight of cage is plotted against the acceleration for various values of n for three cases; in Fig. 8 where there is no strain in the cable ($m = 0$); in Fig. 9 where the slack in the rope is equal to the strain under the weight of the cage ($m = 1$); and in Fig. 10 where the cage is suspended at the moment when acceleration begins ($m = -1$). The straight line A-A in these plates has the equation $\frac{S_{max}}{W} = 1.5 + 0.1f$. It will be noted that it

will give slightly excessive stresses for the suspended weight and safe stresses for ordinary mine hoists, for in these cases n rarely exceeds 10.

As an illustration, the hoisting rope for the mine in which the test was made will be designed. Assume a factor of safety of 4, such as is used in most structural design. Assume also $n = 10$, and $f = 8$ ft. per sec. per sec. The total load would be 21,600 lb. Then stress due to weight and acceleration = 21,600 ($1.5 + 0.1 \times 8$)

$$= 2.3 \times 21,600 = 49,700 \text{ lb.}$$

$$\text{Friction} = (\text{approx.}) \quad 500 \text{ lb.}$$

$$\text{Bending of } 1\frac{3}{4}\text{-in. cable on 8-ft. pulley} = \quad 14,960 \text{ lb.}$$

$$\text{Total} \quad 65,160 \text{ lb.}$$

Working load needed is 65,160 lb. Multiplying by 4 we obtain the necessary proof load in the cable. This is 260,640 lb., or 130.3 tons, which requires a $1\frac{3}{4}$ -in. Monitor plow-steel hoisting rope, $\frac{1}{4}$ in. larger than the rope actually used.

The above bending stress is based on a modulus of elasticity of 12,000,000 lb. per sq. in. This is high, perhaps, for new rope; 16,000,000 lb. per sq. in., however, seems none too high for rope that has had much service. The use of this higher figure would increase the bending stress. However, it may probably be assumed that the factor of safety will provide for any increase in the modulus of elasticity with the use of the rope.

The conclusions reached in this discussion do not necessarily imply that the present design of hoisting ropes makes them unsafe. They do

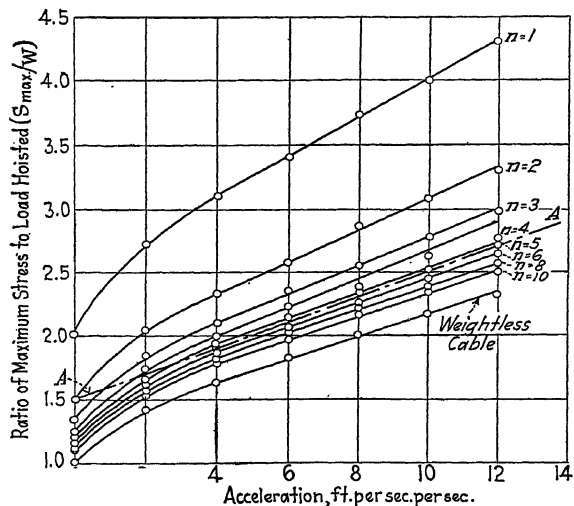


FIG. 8.—RELATION BETWEEN $\frac{S_{max}}{W}$ AND ACCELERATION FOR DIFFERENT WEIGHTS OF CABLE. $M = 0$. NO SLACK OR STRETCH IN CABLE. EQUATION OF DOTTED LINE A-A, $\frac{S_{max}}{W} = 1.5 + 0.1f$.

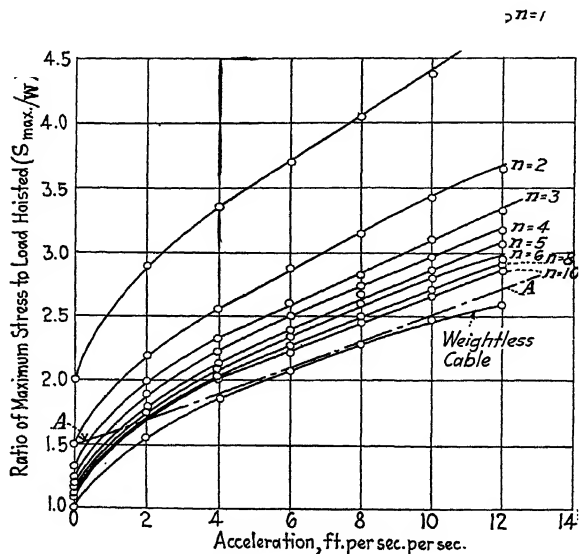


FIG. 9.—RELATION BETWEEN $\frac{S_{max}}{W}$ AND ACCELERATION FOR VARIOUS WEIGHTS OF CABLE WHEN SLACK EQUALS STRETCH CAUSED BY SUSPENDED LOAD. EQUATION OF DOTTED LINE A-A, $\frac{S_{max}}{W} = 1.5 + 0.1f$.

seem to indicate, however, that a slight increase in size of rope should result in a very considerable increase in length of life and that a more rational design will make for a decided increase in ultimate economy.

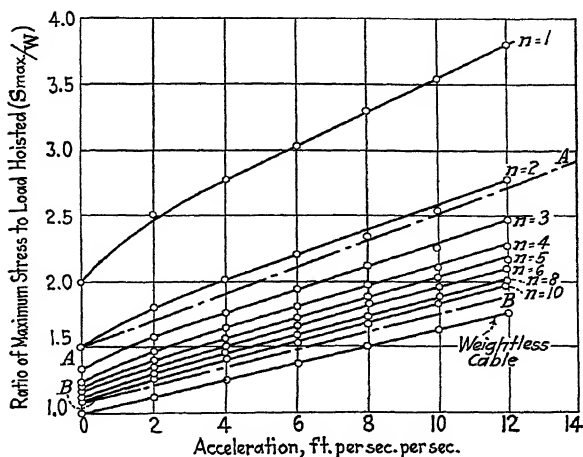


FIG. 10.—RELATION BETWEEN $\frac{S_{max}}{W}$ AND ACCELERATION FOR VARIOUS WEIGHTS OF CABLE WHEN LOAD IS SUSPENDED. $M = -1$. EQUATION OF LINE A-A, $\frac{S_{max}}{W} = 1.5 + 0.1f$. EQUATION OF LINE B-B, $\frac{S_{max}}{W} = 1.1 + \frac{2f}{g}$.

MODULUS OF ELASTICITY OF WIRE ROPE

To ascertain the modulus of elasticity of wire rope, when stressed excessively, the author made extensometer tests on five sizes of 6 by 19 cast-steel wire rope of $\frac{3}{8}$, $\frac{1}{2}$, $\frac{3}{4}$, 1 and $1\frac{1}{4}$ in. diameter respectively.

The extensometers used were of 8-in. gage length. All but the largest cable were tested with a lever type extensometer equipped with a fixed fulcrum and Wissler dial. A Berry strain gage was used on the largest. The gage points could not be placed on the wire rope, so brass rings were soldered to the rope, at distances of 8 in. center to center, from each other and the points set on these in case of the four larger specimens. The points were set in wire rope clamps on the smallest rope.

The rope ends were broomed and coned when received and these cones were set in split wedges hollowed out in conical shape, the split cones having the same taper as the cones on the rope ends. The wedges were set in the machine and the load was applied. The instruments were attached after a slight initial load had been applied.

Loads up to about $\frac{2}{3}$ of the ultimate strength were applied and removed six successive times and the load was carried to rupture on the

seventh application. Stress-deformation readings were observed and recorded on the first and fourth application and removal of load, and on the seventh application.

The cross-sectional area was determined by cutting off a 1-ft. length, removing the hemp core and weighing the steel. Knowing the weight of

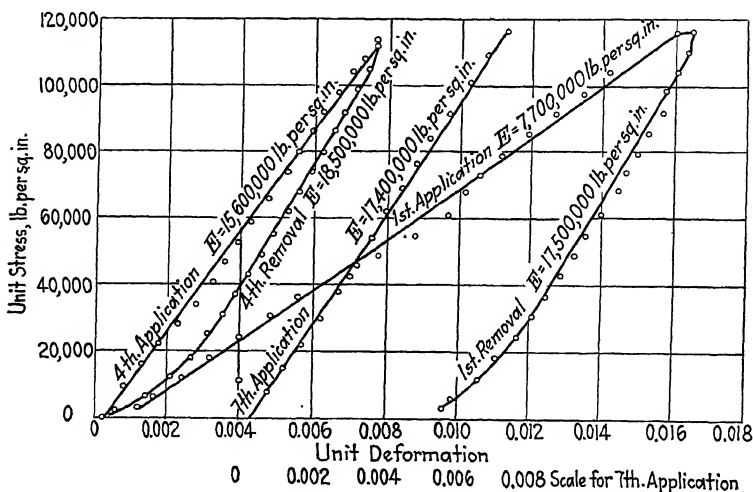


FIG. 11.—STRESS-DEFORMATION CURVES FOR 1¼-in. CAST-STEEL WIRE ROPE USING BERRY STRAIN GAGE. $S_u = 166,400$ LB. PER SQ. IN.

steel per cubic feet, the cross-sectional area was easily obtained. Table 3 gives the results obtained. Fig. 11 is the stress-strain diagram for the 1¼-in. specimen and is typical.

TABLE 3.—*Values of Modulus of Elasticity after Stressing Wire Rope beyond the Elastic Limit*

Rope Diameter, In.	Observed Breaking Load, Lb.	Breaking Load from Catalog, Lb.	Ultimate Strength, Lb. per Sq. In.	Values of Modulus of Elasticity			
				First Application	Average Down	Fourth Application	Seventh Application
¾	8,920	9,600	146,710		16,400,000	14,400,000	
¾	17,100	16,800	155,455	10,500,000	18,300,000	15,000,000	17,200,000
¾	38,275	37,200	158,800	6,000,000	12,400,000		10,400,000
1	69,850	60,000	155,500	6,000,000	18,000,000	16,800,000	14,100,000
1¼	109,000	94,000	166,400	7,700,000	16,500,000	15,600,000	17,400,000

A pronounced twist was noticed in the specimen during the first application of load sometimes, but was hardly noticeable thereafter. This would affect the value of E when the first type of instrument was used, resulting in somewhat smaller value than the actual. The Berry strain gage readings would be affected much less than the others.

ACKNOWLEDGMENTS

The author wishes to acknowledge his indebtedness to Frank Haas, Chief Engineer of the Consolidation Coal Co., as well as his associates, R. L. Kingsland, Chief Mechanical Engineer, and Bruce Grimm, District Mechanical Engineer, without whose cooperation and courtesies the mine-car test could not have been performed.

The wire used to ascertain modulus of elasticity was kindly furnished by the Hazard Wire Rope Co., Wilkes-Barre, Pa.

APPENDIX

DERIVATION OF FORMULA FOR ACCELERATION STRESS IN A WEIGHTLESS ROPE

Assume a weightless elastic rope with a weight, W , attached to its lower end. The upper end moves with a constant acceleration of f ft. per sec. per sec. after motion begins. Let l in. represent the length of the rope. If the weight W were simply suspended, the stretch in the rope would equal $\frac{Wl}{aE}$, in which a sq. in. equals the cross-section of the cable, and E lb. per sq. in. the modulus of elasticity of the rope. Let it be assumed that there is slack in the rope equal to $\frac{mWl}{aE}$ when motion begins, (stretch would be negative slack and could never exceed $\frac{Wl}{aE}$ under ordinary hoisting conditions).

Let α represent the acceleration of the weight in feet per second per second at a time t seconds after the weight W begins to move. Let z equal the stretch in the rope at the same instant. Then $\frac{dz}{dt}$ and $\frac{d^2z}{dt^2}$ represent the velocity and acceleration respectively of the top relative to the weight. Also let T equal the stress in the rope in pounds at this instant.

Then from mechanics

$$T - W = \frac{W}{g} \alpha \quad (1)$$

if the weight of the rope is neglected,

Also $\alpha = f - \frac{d^2z}{dt^2}$, so that

$$T - W = \frac{W}{g} \left(f - \frac{d^2z}{dt^2} \right) \quad (2)$$

Again from mechanics,

$$T = \frac{zaE}{l} \quad (3)$$

Substituting (3) in (2) and rewriting,

$$\frac{d^2z}{dt^2} + \frac{zgaE}{Wl} = f + g \quad (4)$$

From differential calculus, the general solution of (4) is

$$Z = C_1 \cos (A. t) + C_2 \sin (A. t) \quad (5)$$

$$\text{in which } A = \sqrt{\frac{gaE}{Wl}} \quad (6)$$

and C_1 and C_2 are general constants.

The particular solution is

$$Z = \frac{Wl}{gaE}(f + g) \quad (7)$$

Then the complete solution is the sum of (5) and (7), or

$$Z = C_1 \cos (A. t) + C_2 \sin (A. t) + \frac{Wl}{gaE}(f + g) \quad (8)$$

differentiating

$$\frac{dz}{dt} = -AC_1 \sin (A. t) + AC_2 \cos (A. t) \quad (9)$$

Time is reckoned from the instant when the weight begins to rise. At this instant the stretch in the rope = $\frac{Wl}{aE}$. That is when $t = 0$; $z = \frac{Wl}{aE}$. Also the top has moved a distance equal to the slack $\frac{mWl}{aE}$ plus the stretch $\frac{Wl}{aE}$, needed to suspend the cage or a distance $\frac{Wl}{aE}(m + 1)$. Now $S = \frac{v^2}{2f}$, the distance moved from rest by a uniformly accelerated motion. In this instance $V^2 = \frac{2fWl}{aE}(m + 1)$ or $V = \sqrt{\frac{2fWl}{aE}(m + 1)}$, in which V is the velocity of the top when the weight begins to rise. This is also the velocity of the top relative to the weight when $t = 0$ and, as previously shown, is $\frac{dz}{dt}$.

$$\text{Hence when } t = 0, \frac{dz}{dt} = \sqrt{\frac{2fWl}{aE}(m + 1)}.$$

Substituting $t = 0$ and $z = \frac{Wl}{aE}$ in equation (8)

$$\text{and } t = 0 \text{ and } \frac{dz}{dt} = \sqrt{\frac{2fWl}{aE}(m + 1)} \text{ in equation (9)}$$

and solving

$$C_1 = -\frac{Wlf}{gaE} \quad (10)$$

$$C_2 = \frac{Wl}{aE} \sqrt{\frac{2f}{g}(m + 1)} \quad (11)$$

The complete solution with constants evaluated becomes

$$Z = -\frac{Wlf}{gaE} \cos (A. t) + \frac{Wl}{aE} \sqrt{2f(m + 1)} \sin A. t + \frac{Wl}{gaE}(f + g) \quad (12)$$

and its first derivative:

$$\frac{dz}{dt} = \frac{AWl}{aE} \left[f \sin A. t + \sqrt{\frac{2f}{g}(m+1)} \cos A. t \right] \quad (13)$$

Now z is a maximum when $\frac{dz}{dt} = 0$. Hence, equating (13) to zero, solving for $\sin A. t$ and $\cos A. t$, and substituting in (8)

$$Z_{max} = \frac{Wl}{aE} \left[1 + \frac{f}{g} + \frac{\sqrt{f^2 + 2fg(m+1)}}{g} \right].$$

Now stress is proportional to stretch. The stretch due to the weight

$$W = \frac{Wl}{aE}$$

$$\text{Then } \frac{S}{W} = \frac{\frac{Wl}{aE} \left[1 + \frac{f}{g} + \frac{\sqrt{f^2 + 2fg(m+1)}}{g} \right]}{\frac{Wl}{aE}},$$

$$\text{or } S_{max} = W \left[1 + \frac{f}{g} + \frac{\sqrt{f^2 + 2fg(m+1)}}{g} \right],$$

which is equation (1).

DERIVATION OF THE GENERAL EXPRESSION FOR ACCELERATION STRESS IN HOISTING ROPES

Let l = length of rope, ft.

W = weight hoisted, lb.

W' = weight of rope, lb.

f = constant acceleration at top of rope.

α = variable acceleration of load hoisted.

T = tension in rope at top, lb.

T' = tension in rope at load, lb.

Z = stretch in whole length of rope, in.

a = cross-sectional area of rope, sq. in.

E = modulus of elasticity of rope, lb. per sq. in.

Then $\frac{\alpha + f}{2}$ = mean acceleration of rope, $\frac{dz}{dt}$ = velocity of top relative to load and $\frac{d^2z}{dt^2}$ = acceleration of top relative to load. Also, $\frac{d^2z}{dt^2} = f - \alpha$.

The rope stretch Z , at any instant may be divided into four components; $\frac{Wl}{aE}$ due to weight of load, $\frac{W'l}{2aE}$ due to weight of rope, $\frac{W\alpha l}{gaE}$ due to acceleration of load, and $\frac{W'l}{2gaE} \cdot \frac{f + \alpha}{2}$ due to acceleration of rope.

$$\text{Thus } Z = \frac{l}{aE} \left[W + \frac{W'}{2} + \frac{W\alpha}{g} + \frac{W'}{4g}(f + \alpha) \right] \quad (1)$$

Now $\alpha = f - \frac{d^2z}{dt^2}$, whence by substitution

$$Z = \frac{l}{aE} \left(W + \frac{W'}{2} + \frac{Wf}{g} - \frac{W}{g} \frac{d^2z}{dt^2} + \frac{W'f}{2g} - \frac{W'}{4g} \cdot \frac{d^2z}{dt^2} \right) \quad (2)$$

Now let $W' = \frac{W}{n}$ and substitute in (2)

$$Z = \frac{Wl}{2gnaE} \left[2gn + g + 2fn + f - \frac{4n+1}{2} \cdot \frac{d^2z}{dt^2} \right] \quad (3)$$

or
$$\frac{d^2z}{dt^2} + \frac{4gnaE}{(4n+1)Wl} \cdot Z = \frac{2(f+g)(2n+1)}{4n+1} \quad (4)$$

The general solution of this differential equation is $Z = C_1 \cos At + C_2 \sin At$ in which $A = \sqrt{\frac{4gnaE}{(4n+1)Wl}}$ and C_1 and C_2 are general constants.

The particular solution is $Z = \frac{Wl}{2gnaE} (f+g)(2n+1)$.

The complete solution is

$$Z = C_1 \cos At + C_2 \sin At + \frac{Wl}{2gnaE} (f+g)(2n+1) \quad (5)$$

Reckon time from the instant when the load begins to move upward. At this instant

$$Z = \frac{Wl}{aE} + \frac{Wl}{2naE} + \frac{Wlf}{6gnaE} \quad (6)$$

Then when $t = 0$; Z has the above value.

Substituting in (5)

$$C_1 = -\frac{Wlf}{3gnaE} (3n+1) \quad (7)$$

Assume an initial slack in the rope of Z_0 . Then when $t = 0$, the top will have moved a distance equal to $Z_0 + \frac{Wl}{2gnaE} \left(2gn + g + \frac{f}{3} \right)$ with a constant acceleration of f and its velocity will then be equal to

$$\sqrt{2f \left(Z_0 + \frac{Wl}{2gnaE} \left[2gn + g + \frac{f}{3} \right] \right)}.$$

This is the value of $\frac{dz}{dt}$ when $t = 0$.

C_2 may now be evaluated by substituting the above value for $\frac{dz}{dt}$ and zero for t in the derivative of Eq. (5). By so doing,

$$C_2 = \frac{\sqrt{2f \left[Z_0 + \frac{Wl}{2gnaE} \left(2gn + g + \frac{f}{3} \right) \right]}}{A} \quad (8)$$

The complete solution of (4) with constants evaluated then becomes

$$Z = -\frac{Wlf}{3gnaE}(3n+1)\cos At + \frac{\sqrt{2f\left[Z_0 + \frac{Wl}{2gnaE}\left(2gn + g + \frac{f}{3}\right)\right]}}{A} \sin At + \frac{Wl(f+g)(2n+1)}{2gnaE} \quad (9)$$

Now the stretch Z is greatest when $\frac{dz}{dt} = 0$. Solving for $\cos At$ and $\sin At$ in the first derivative of (9) when equated to zero, and substituting these values back in equation (9) as well as replacing Z_0 by $\frac{mWl}{aE}$, the expression for Z_{max} is obtained

$$\frac{Wlf}{gnaE} \left[\frac{2\left(\frac{3n+1}{3}\right)^2 \sqrt{\frac{f}{4n+1}} + \left(2gmn + 2gn + g + \frac{f}{3}\right) \sqrt{\frac{4n+1}{4f}}}{\sqrt{2gmn + 2gn + g + \frac{f}{3} + \frac{4f}{4n+1} \left(\frac{3n+1}{3}\right)^2}} + \frac{Wl}{2gnaE}(f+g)(2n+1) \right] \quad (10)$$

$$\text{At any instant } T = W + \frac{W}{n} + \frac{W\alpha}{g} + \frac{W(f+\alpha)}{2gn} \quad (11)$$

and is obviously greatest when α is a maximum. But $\alpha = f - \frac{d^2z}{dt^2}$ and will be greatest when $\frac{d^2z}{dt^2}$ is a minimum.

$$\text{From (4)} \quad \frac{d^2z}{dt^2} = \frac{2(f+g)(2n+1)}{4n+1} - \frac{4gnaEz}{Wl(4n+1)} \quad (12)$$

and is a minimum when Z is a maximum.

Hence, substituting for α in (11) according to this reasoning

$$T_{max} = W \left(1 + \frac{1}{n} + \frac{f}{2gn} + \frac{2n+1}{4n+1} \left[\frac{f(4n+1)}{2gn} - \frac{2(f+g)(2n+1)}{2gn} + \frac{2Z_{max}}{\frac{Wl}{aE}} \right] \right) \quad (13)$$

This equation, after substituting for Z_{max} its value in (10) was used for calculations shown in Table 1 and Figs. 8, 9 and 10.

DISCUSSION

C. E. LAWALL, Morgantown, W. Va.—The equations which Professor Boomsliter shows in Figs. 8, 9 and 10 seem to indicate that the equation adapted to the rope when it has no slack or stretch is the one that is most representative. Would it be permissible to use this equation in most cases?

G. P. BOOMSLITER.—How much slack or stretch there is in the rope when the load is being hoisted is a question. We know that when the loaded cage goes into

the dumping horns, there is slack in the rope; therefore it will begin to rise with slack in the rope. On the other hand, I think careful engineers have allowed a small interval of time for taking up this slack. If the engineer waited long enough for the vibration to cease after taking up the slack the condition would be the same as for ordinary elevators in buildings; in other words, as for a suspended cable. However, the period for taking up the slack is very short, perhaps a second or two, and it is probable that the vibration has not entirely ceased. The hoisting engineer gets the coal out just as fast as he can and will make this period short; therefore it would seem reasonable to assume that we have neither the case of the suspended load on the one hand nor that of a considerable amount of slack on the other, particularly if we try to take up the slack before we begin to hoist. It seems most reasonable to assume the curves of Fig. 8, in which there is neither slack nor stretch in the cable.

F. L. STONE, Schenectady, N. Y.—Is the vibration in Fig. 5 continued during the entire cycle?

G. P. BOOMSLITER.—That is a hoisting period. Fig. 6 shows a complete cycle.

F. L. STONE.—Do you anticipate that the vibration would be reflected back in the motor? If so, it would produce corresponding current fluctuations.

G. P. BOOMSLITER.—I do not see how it could help it.

F. L. STONE.—I have taken oscillograms on a very fast hoist of the Old Ben Coal Co. with steep cones or scrolls. The hoist is considerably faster than the one under discussion and the records show nothing of the effect of rope vibration. This, I believe, is accounted for by the fact that these vibrations are damped out by virtue of the catenary from the sheave wheel to the drum and also by the inertia of the sheave wheel itself.

Generally speaking, the only way to make the rope stronger is to make it larger. One of the difficult problems of mine hoisting is the size of the rope. The moment you increase rope diameter, you must make a corresponding increase in the drum diameter, which in turn increases the inertia or WR^2 , and so on. The ratio of rope diameter to drum diameter should not be less than 60:1.

G. P. BOOMSLITER.—You will see very little of that, I take it, for this reason: The drum is a large revolving mass, the inertia of which is very great, and since this is revolving as the cage rises the effect of the cord will be relatively small, as it would not appear to be reflected.

The design which I have given only increased the size of the cable from $1\frac{1}{2}$ to $1\frac{3}{4}$ in. to get what seemed like a fair factor of safety.

F. L. STONE.—On page 86 in the tabulation of the ratio of maximum stress to weight, I note that when you lift the loaded car and cage, the observed almost invariably, I think with one exception, is under the calculated, according to the formula, and I would like to know if it was closer to the common theory than the formula.

G. P. BOOMSLITER.—My own interpretation is that it was not; that it was nearer to the formula. It will not be the same largely because we have friction and the dampening effect of the air in the shaft.

Treading across these three columns for all trips, and neglecting trip No. 2 up, Series 1, in which the circuit breaker went out, 11 out of 17 observed stresses were nearer to Formula 1 than to the common theory.

F. L. STONE.—You have one acceleration of 11.25 ft. per sec. per sec., which is exceedingly high.

G. P. BOOMSLITER.—That was one of those jerky conditions.

Physiological Effects of Mine Dusts

By EDGAR L. COLLIS,* CARDIFF, WALES

(New York Meeting, February, 1927)

No industry or group of industries is more deeply interested in the influence exerted by atmospheric dust than that concerned with the getting of coal and of metalliferous ores. The coal miner in the past has mainly been interested on account of the part played by coal dust in the occurrence of explosions; but investigation is revealing that coal miners experience, at least on certain coal fields, unduly high mortality ascribed to lung diseases, some part of which may reasonably be attributed to dust inhalation, and that a small group of men employed in sinking shafts and driving roads through sandstone rocks experience troubles identical with those that afflict many metalliferous miners.

COAL MINING

The Registrar General for England and Wales issues information every 10 years concerning the mortality experienced in various industries. The records dealing with coal mining are of particular interest, since they are given separately for each of the largest fields. When examined they exhibit the coal miner with a favorable death rate on the whole when compared with all other occupied males; but they reveal great differences on the different fields; thus the Lancashire miner in 1900-2 had a comparative mortality from all causes of 1006 as against a figure of 675 for the Nottinghamshire and Derbyshire miners, and in 1910-12 of 941 as against 570 for Nottinghamshire. When mortalities due to four only of the great causes of death are subtracted from the total the differences are greatly reduced; these four causes are phthisis, pneumonia, bronchitis and accidents. Thus for 1910-12 more than one-half of the mortality in Lancashire, 478 out of 941, is so accounted for, but less than one-third of the mortality in Nottinghamshire, 184 out of 570; the latter fraction compares with that for all males, 293 out of 790. Exactly why the accident mortality varies thus with those for the lung diseases has not been determined; the fact, however, points to an interesting association between health and so-called "accidental" occurrences.

* Talbot Professor of Preventive Medicine, University of Wales; member of Health Advisory Committee and Miners' Welfare Committee, British Ministry of Mines.

One point which stands out is that the mortality from phthisis, although varying widely on the different coal fields is low on all. Low though it is, it possesses certain characteristics; it is high where mortality from other lung diseases is high and its maximum incidence occurs later in life, at ages 55 to 64, than in other occupations. These characteristics are peculiar to underground work; and phthisis as it occurs among men working on the surface at the pithead does not possess them; for instance the maximum incidence of phthisis among workers above ground, 1910-12, was at ages 20 to 24. They are characteristics which are peculiar to dust-phthisis, or silicosis, and they pointed to the existence of this disease in the mining community before clinical investigations disclosed its existence among that small group of underground men who are especially exposed during their work to silica-dust. The exposure is associated with rock work and is becoming more and more intensive as percussive drills or jack-hammers are being generally introduced for this work.

TABLE 1.—*Comparative Mortality among Coal Miners, Ages 25 to 64, Inclusive*

Field	1910-1912						Phthisis	Pneumonia	Bronchitis	Accidents
	All Causes	Phthisis	Pneumonia							
Nottinghamshire*....	570	53	40	25	66	67	64	52	49	80
Derbyshire*.....	591	70	34	39	73					
Northumberland and Durham.....	635	70	54	33	83		84	54	41	105
Staffordshire.....	717	74	70	61	109		66	71	104	118
Yorkshire.....	758	81	69	45	117		88	71	67	99
Wiltshire and South Wales.....	777	70	69		131		93	108	104	169
Lancashire.....	941	107	100		183		96	149	113	131
All coal fields.....	727						89	86	79	123
Occupied and retired males.	790	142	66	38	47		187	92	58	58

* Nottinghamshire and Derbyshire were grouped together previous to 1910-1912.

Coal dust does not in itself appear to exert any particularly harmful effect on the lungs, as may be seen by comparing the mortality from lung diseases experienced by the miners of Nottinghamshire and Derbyshire with that of all occupied males. Possibly the low phthisis mortality among coal miners is sufficiently explained by the isolation of the men underground when at work which does not favor easy transmission of the disease by direct infection; in this respect the miner is somewhat

similarly placed to the agricultural laborer whose mortality from this disease is similarly low.

TABLE 2.—*Mortality per Thousand from Phthisis for Three Decennial Periods among Persons Employed in Certain Occupations and at Various Age Periods*

Occupation	Periods	Age Period						
		15	20	25	35	45	55	65 and Over
Coal miners.....	1890-92	0.58	1.39	1.21	1.45	2.07	2.23	2.03
	1900-02	0.41	1.03	0.96	1.09	1.52	2.04	1.47
	1910-12	0.54	1.81	0.84	1.02	1.31	1.43	1.02
Tin and lead miners.....	1890-92	1.10	3.21	3.78	6.15	10.58	13.85	13.89
	1900-02	0.56	1.54	6.01	7.85	10.50	12.22	10.00
	1910-12	0.20	2.31	3.24	9.23	10.44	13.72	8.24
Printers.....	1890-92	1.42	4.21	4.97	5.87	5.22	5.30	4.39
	1900-02	1.03	3.45	3.65	5.13	4.34	3.68	1.87
	1910-12	0.78	2.60	2.64	3.07	3.37	2.90	2.19
Inn servants.....	1890-92	0.42	2.73	6.01	11.15	8.65	3.95	2.40
	1900-02	0.80	2.20	5.49	9.99	9.65	5.48	6.51
	1910-12	0.57	1.26	3.13	5.02	4.91	3.85	1.50
Agricultural laborers.....	1890-92	0.41	1.41	1.83	2.11	1.87	1.60	0.95
	1900-02	0.30	1.05	1.15	1.29	1.43	1.26	0.75
	1910-12	0.28	0.82	1.00	1.10	0.98	0.87	0.61
All males.....	1890-92	1.11	2.15	2.76	3.48	3.47	2.86	1.44
	1900-02	0.82	1.69	2.16	2.89	3.15	2.52	1.33
	1910-12	0.74	1.38	1.70	2.09	2.29	2.17	1.14

Stone Dusting

The known harmful influence exerted by silica dust is of interest in connection with stone-dusting the galleries and roadways of coal mines to prevent explosions. No dusts which contain silica should ever be used for this purpose. Personally I only feel safe as regards the use of limestone dusts for this purpose; they at any rate are harmless when inhaled and they possess the additional advantage, due to their white color, of lightening up the roadways, no small advantage in British coal mines where that distressing occupational disease, miners' nystagmus, is prevalent, the etiology of which is so closely bound up with the question of illumination.

Many other dusts are used for stone-dusting. They can not be condemned as possessing the peculiar dangers of silica, nor can they be

recommended as being equally harmless with the lime stones. They may possess potentialities with regard to the prevalence of pneumonia and bronchitis. Fortunately, exposure to inhalation of these dusts is small, except possibly for those who carry out the actual work of dusting. Should these men develop a tendency to suffer unduly from these lung diseases, then the dust used will have to be changed. Some of the shales used even contain appreciable amounts of silica, but there is evidence that the dust of coal measure clays, which are usually present in shales, when intimately mixed with silica dust exerts an inhibiting influence on silica.

Health and Psychology

Attention may be drawn before leaving the coal miner to a correlation found between his ill health, as measured by mortality, and industrial discontent, as measured by ballots taken of miners in Great Britain in connection with labor disputes. The figures of these ballots may be regarded as direct indications of the extent to which dissatisfaction prevails on the different coal fields. The results are shown in Table 3; the orders of the fields so given on the three occasions are curiously alike. Lancashire always leads and South Wales is always second, while Nottinghamshire (with the exception of 1920 noted below) stands on each occasion at the bottom. The similarity of the orders as obtained by the ballots, and the close similarity found (see Table 1) between them and the orders as obtained by mortality records, can hardly be chance occurrences.

TABLE 3.—*Results of Strike Ballots of the Miners Taken in Different British Coal Fields*

Coal Field*	Percentage of Miners That Voted		
	For Strike August, 1920	Against Re- suming Work June, 1921	Against Wage Agreement January, 1924
Nottinghamshire.....	55.1	53.7	69.2
Derbyshire.....	71.8	52.7	76.8
Northumberland and Durham.....	69.9	66.7	75.9
Yorkshire.....	51.1	65.3	75.1
South Wales.....	77.9	73.0	89.8
Lancashire.....	90.6	89.6	94.3

* Separate ballot figures were not published for Staffordshire, so that composite field is excluded.

One interesting apparent exception in the order of 1920, according to ballot, should be noted: Yorkshire is clearly out of the position which mortality records ascribe to it, but in this field a strike occurred in 1919 which did not involve other coal fields, and the effect of this event was

held to modify the votes cast for the strike in 1920. In 1921 and 1924 when all the coal fields started from the same basis, the percentage vote indicative of dissatisfaction places Yorkshire much more nearly in the order according to mortality records for 1910 to 1912. Here is found evidence that matters of health influence the mind of the miner, and are at the back of that industrial unrest which tends too frequently to express itself in the economic disaster of a strike. The way to remedy this psychology, with its results so paralyzing to the community, is to alter the conditions which give rise to it so that ill-being and ill-health are reduced to a minimum.

METALLIFEROUS MINING

Probably the occurrence of silicosis among coal miners as a whole would have passed undetected had not study of this disease in metalliferous mining and also in such other industries as the grinding of metals, the dressing of sand stones and of granite, and the manufacture of pottery, disclosed the characteristics of silica dust as a death-dealing hazard. The extent to which this hazard claims its victims may be seen from the mortality experienced by British tin and lead miners when compared with other occupations, such as those of printer and inn servant, both representative of occupations with a high mortality from phthisis, that of agricultural labor with a low mortality, and that of all males taken as the standard.

Some interest attaches to what takes place in the lungs when silica dust is inhaled. First let me say that the dust must be small enough to pass through the finest of the air passages into the terminal cavities or alveoli of the lungs; such particles vary in size from 1 to 5 microns, or micromillimeters, the great majority being about 1 micron in diameter. From these cavities the particles are picked up by certain large cells, macrophages, which engulf or eat them. Some of these cells burdened with the dust find their way out up the air passages in sputum but others pass into the lung tissue through the walls of the alveoli and become the source of trouble. The silica particles dissolve in the tissues and poison them, definitely causing necrosis or local death. Some idea of what occurs can be obtained by examining X-ray photographs of the chest at different stages of the process. At first the air passages may be noted branching out and dividing like the boughs and twigs of a tree; so far the picture resembles what may be seen in the lungs of any city dweller. Then a new thing comes, little spots appear like the leaves on the twigs, or snow flakes in a storm. These spots represent changes going on in the lungs which become altered thereby from a spongy consistency to a tough fibrous mass; the end result is known as fibrosis. Lungs in this condition do not function well; the worker is unable to breathe deeply; he becomes short of breath on exertion. The condition may advance

without any complication to a fatal issue. The process of solution of silica in the body seems to be a slow one. It may progress, or at least the fibrosis it causes may progress, for some years after all exposure to dust inhalation has ceased. Silica dust possesses, in addition to this tendency to set up fibrosis in the lungs, characteristics common to other dusts, such as those of emery, carborundum and other inorganic materials, which do not naturally form part of living matter. Such dusts irritate the air passages and cause bronchitis and, if they are small enough to get into the smallest air passages and alveoli, pneumonia as well. Indeed the chief of all the diseases caused by dust is bronchitis, chief on account of the lost time it causes due to sickness and on account of the mortality it originates.

SILICOSIS

The fibrosis caused by silica is, however, something distinctive, which can be recognized, just as can lead poisoning, as a hazard peculiar to industry, hence much attention has been concentrated thereon. Simple uncomplicated fibrosis may advance to a fatal issue, but in fact it is more often complicated by the intervention of tuberculous infection. Exactly why tissues damaged by silica fall a ready prey to the tubercle bacillus is not known. Gye and Kettle have shown that whether silica be injected into the tissues of animals, or hydrated soluble silica, the result is the same, the action only being slower if the powder form is used. An animal can be killed merely by placing a sufficient amount of silica in its tissues. It is in fact a definite poison.

Kettle has further demonstrated that, if the tissues are damaged by such material as calcium and also by silica and if tubercle bacilli are then subsequently injected, the bacilli multiply and increase in far vaster numbers at the lesions caused by the silica than at those set up by calcium or by the other agents.

Here is a rough demonstration of what takes place when a human being with silicotic fibrosis of his lungs becomes exposed to tuberculous infection. If this form of infection can be rigidly excluded then the most serious effects of silicosis will be avoided. On such lines, among others, action is taken on the South African gold fields where strenuous endeavor is made to exclude tuberculous persons from employment by preliminary medical examinations and by periodical examinations thereafter. An interesting point is that the supervision of tuberculous infection appears to precipitate a previously latent silicosis.

DUST PREVENTION

Useful, however, as is such exclusion of tuberculous infection from those exposed to a silica dust hazard, of greater import is it to reduce to

a minimum the dust hazard itself. In mining, dust is generated in three ways, (1) in drilling holes, particularly when percussive drills are used, (2) in shot firing, and (3) in breaking up and removing rock. The dust hazard can be reduced in various ways, (a) by keeping material damp so that dust is not formed, (b) by preventing any dust generated from gaining access to the atmosphere, (c) by allowing dust generated to settle out of the air before anyone enters it, and (d) by the use of respirators of various kinds.

Damping may be of some use when breaking up and removing rock, but here the use of respirators may have to be resorted to; water has been used in drilling, but has not proved entirely successful for suppressing dust.

The only way of dealing with the dust generated by shot firing is to allow a sufficient interval to elapse after firing before men enter the air. For this purpose firing should only take place at the end of a shift. From 2 to 3 hr. are required in a still atmosphere for the fine dangerous particles of dust to settle sufficiently.

The dust generated during drilling has always been the most difficult problem, but recently a device has been worked out which gives promise of solving it at least for the percussive drill. Nothing has yet been devised for hand workers, such as the stone mason; possibly his salvation will be found in introducing pneumatic tools for all stone cutting and carving. In the device referred to the air escaping from the percussive drill is made to discharge lengthwise through a nozzle into the middle of a tube; this escaping air, acting on the ejector principle, causes an inflow of air into the open end of the tube; this open end is placed around the shot hole and the inflowing air picks up the dust generated and prevents its escape into the atmosphere. The far end of the tube is terminated in the top of a bag-filter made of ordinary flannel. The bottom of the filter bag is held by an elastic band to a metal receiver. The incoming air passes through the flannel leaving the dust within. As the blows of the hammer release the air in puffs, the exhaust draught comes into the bag-filter in puffs which shake the dust free to fall into the receiver.

Dust counts made of the air escaping from the bag-filter show a content well below the 300 particles per c. c. permitted in the South African mines. Since, however, the filter can be placed several yards away from where the man is at work, the dust content of the air he breathes should be well below the danger threshold.

In order to prevent exploitation of the principle concerned with this device a patent has been taken out by the British Department of Mines, but with no intention of interfering with anyone who uses it. The principle concerned seems sound, as the more the tool works, the greater is the exhaust draught created. In application it is simple; but as it comes into use improvements are certain to be introduced.

DISCUSSION

R. R. SAYERS, Washington, D. C.—The apparatus described by Dr. Collis was developed by Capt. P. S. Hay.¹ He has been in correspondence with Philip Drinker² and G. S. Rice of the Bureau of Mines, suggesting to the latter the study of it in connection with coöperative work being carried on between the U. S. Bureau of Mines and the Safety in Mines Research Board of England. This work is now under consideration.

The apparatus does not have to be applied tightly against the rock face. Some of the apparatus previously developed, where water was used as in the Carroll dust collector, had to be applied tightly against the rock face. This was very difficult to do, but in suction apparatus it is unnecessary.

A number of years ago in the course of studies along this line in South Africa, an attempt was made to use the exhaust from the mining machine. Dr. Mavrogordato has very recently made a statement that it is not practicable to use the exhaust air from mining machines in South Africa to collect the dust, and they have been using air from the high-pressure side and find it more satisfactory.

W. WRIGHT, New York, N. Y.—Dr. Dublin can, perhaps, offer some comment relative to the distinction between bronchitis as recognized in the British statistics, and bronchitis as recognized in this country, because they are two different things.

L. I. DUBLIN, New York, N. Y.—In Great Britain what we call chronic bronchitis is very much more prevalent than in our country. It has seemed to me that what we call tuberculosis is often called chronic bronchitis on the other side. I got into correspondence with Dr. Stevenson, and while he agreed that that was partly true, nevertheless he insisted that there was a very great prevalence of chronic bronchitis in England, much more so than here, and ascribed it to the peculiarities of the climate.

I do not know whether that explains the situation for these dusty trades. My impression is that one cannot make a sharp distinction between tuberculosis and chronic bronchitis, in a great many instances, and that perhaps the safer way to compare the situation on the other side with our own here would be to combine chronic bronchitis and phthisis as one entity, knowing full well that we are not dealing with a single thing. For purposes of comparison, it would be safer to get the addition of these two conditions.

The picture that Dr. Collis has drawn for his country is also the picture for our own. The relationship between the tuberculosis of coal miners and of other miners to the general population mortality from tuberculosis is quite identical. We also show a uniformly low tuberculosis rate among our coal miners and very high tuberculosis rates among metal miners and those who are exposed to the hard stone dust.

W. B. MCKINLAY, Yonkers, N. Y.—Dr. Collis, would there be included in these mortality figures deaths of miners who may have ceased active work in the mines for a long period before they died? During a year's residence in Colorado I heard of specific cases, perhaps six or seven, where men died during the year of what the doctors said was silicosis. Every one of these men had quit work as a miner from 3 months to 5 years before death. I do not think any of these deaths will appear in statistics on such deaths. The inference is that the danger is greater than the available statistics show.

E. L. COLLIS.—Considerable care is taken in Great Britain to get the deaths and the occupation of the individual properly recorded.

¹Scientific Executive Officer, Safety in Mines Research Board, Mines Department, England.

²Assistant Professor of Ventilation and Illumination, Harvard School of Public Health.

W. B. MCKINLAY.—I do not think our statisticians are so careful; or rather, they do not have access to the facts.

E. L. COLLIS.—The information is particularly difficult to obtain. I presume you would have more difficulty than we have in relation to that matter.

With reference to the point that Dr. Dublin raised, it is one that I started out on for the purposes that he suggested, but I gave it up because of the position of people like our shoemakers and printers who have a high phthisis rate but low bronchitis and pneumonia rates. If I were to combine the three diseases, I would be unable to get my statistical differential diagnosis between the groups. I am quite sure that the big peak of bronchitis does represent a very definite clinical entity which is not phthisis, although it may be exaggerated. Keeping them separate according to the industries helps differential diagnosis and statistics.

A. M. TWEEDY, New York, N. Y.—Can fibrosis or silicosis be cured in any way? Will time cure it?

E. L. COLLIS.—The condition advances even after removal from risk. The process of passing into solution and attacking the tissues of the silica particles appears to take considerable time. In South Africa they X-rayed men before they went into the service in the war and when they came back 2 or 3 years later (they had had no exposure to risk); the condition of their chests had advanced considerably, so that from the point of view of what will happen from silica dust, the process goes on after exposure to risk has stopped.

Tuberculosis is a separate disease which may come on at any time.

R. R. SAYERS.—I would like to amend that slightly. In South Africa there are occurrences such as Dr. Collis has described. In other cases there can be improvement. Sanitariums have been established there and cases even in the secondary or late stages have improved very materially. They do not get well. It is claimed that in the very early stage, the anti-primary stage, the man can improve until he is practically back to normal efficiency. It is not claimed that the lung tissue has returned to its original state.

J. B. PORTER, Montreal, Que.—Dr. Collis mentioned the fact that silicosis does not necessarily originate with actual silicate dust but may be artificially induced, let us say, by injections of soluble silica. Would the injections have to be intravenous or is it conceivable that they might be injections of soluble silicates through the alimentary canal? Another question I have in mind relates to the beginning of Dr. Collis' paper where he points out a very interesting parallelism between certain diseases among the miners. Has the information regarding the recent coal strike come far enough to show whether there is a parallelism between the different coal fields?

E. L. COLLIS.—The silica in each of the instances I gave was injected subcutaneously by Professor Kettle. What would happen if gotten into the alimentary tract raises certainly an extremely interesting point. Ordinary glass may contain a certain amount of free silica. It is possible that if fluid lay sufficiently long in a glass it might be dissolved out.

As regards the recent strike, unfortunately I could not get any data; no ballot was taken.

L. I. DUBLIN.—I was very much struck with what Dr. Collis said and the work of others in regard to the very long duration of some of these cases of individuals exposed to silica dust. It is not at all uncommon to find workers in the quarries who have been at it 25 to 30 and more years. They are just as much exposed as others who succumb after 2 or 3 years. That suggests that there is an individual idiosyncrasy

there; that individuals taken by and large vary very much in their ability to withstand heavy exposures to silica dust.

Does it not suggest the possibility or rather the desirability of an investigation into the types of individuals who succumb quickly and more slowly to these exposures? That would be a very important investigation if it could be carried out to a solution, because obviously it would pay the industry to select individuals who would ordinarily have long life apart from what other measures might be taken to minimize the dust hazard.

There may well be some specific physical characteristic correlated with ability to withstand exposure to these dangerous dusts.

E. L. COLLIS.—It is a very interesting point. Others working from an entirely different aspect, knowing nothing of South African work, declare that this cylindrical or vertical heart is associated with what they call the alkaline group of humanity, or rather with people whose acidity is less; and men working from an entirely different angle say that these people are more liable to tuberculosis. They knew nothing of South African work at all.

It is also interesting to note that the mortality records nearly always show 10 years' work (stonemason particularly) before there is any rise in the phthisis death rate. The mason is below the normal for phthisis up to age 25, and yet he is exposed to this silica risk, which seems to be a specific risk which lowers the resistance of the body to tuberculosis. Then why have they got to work 10 years in an industry before they begin to show the effects of it? I know it is more rapid than the intense exposure of the jackhammer, but where you have got it sufficiently spread out as with the stonemason, there is a 10-year period before the phthisis rate gets up high. It is quite an interesting point.

W. WRIGHT.—The fact that silicosis is related to our own coal mining industry is worthy of some comment. I do not know regarding the condition in England, but silicosis is looming up as a matter of great importance in connection with our anthracite mining.

E. L. COLLIS.—There is only a small number of coal miners concerned, those who do the drilling in hard rocks. There are more than a million miners, the vast majority of whom are only getting coal, but the silicosis of the small group lifts the mortality of the whole at late middle life. We have had a certain number of this group X-rayed, and there is no question but that silicotic tuberculosis is present, and they die in just the same numbers as metalliferous miners, if there is a sufficient number of them employed in that work.

L. I. DUBLIN.—Yes, and a great many have already been taken out of the picture through accidents.

E. L. COLLIS.—Yes.

R. R. SAYERS.—I might say that we have found in going over the statistics that the respiratory diseases taken as a whole among coal miners are more numerous than among the general population. The death rate due to tuberculosis among coal miners is lower than in the general population so far as we can find from the death certificates.

We have found on the examination of a number of coal miners that their lungs do show fibrosis, but nothing like the amount of fibrosis that we will find among the hard rock miners.

We know that there are a number of different types of silicosis. There is one known as the Broken Hill type where the appearance is described as soft. The South African type is described as a sharply defined, hard formation. It is all very interesting but needs more study as Dr. Collis suggests.

Underground Air Conditions and Ventilation Methods at Tonopah, Nev.*

By B. O. PICKARD†, BERKELEY, CALIF.

(New York Meeting, February, 1927)

WITH more than a score of shafts and numerous stope openings to the surface, all inter-connected underground; with underground temperatures high, often exceeding 100° wet bulb; with an ore presenting a dust health-hazard; with certain rocks giving off inert gases; with an altitude of 6000 ft. above the sea level and a dry desert climate, the Tonopah mines offered an interesting problem to the student of metal-mine ventilation.

Tonopah is not an old mining district, relatively speaking. It was discovered in 1900 and Tonopah soon was a boom camp. By 1910 the mines of this district had paid many millions of dollars in dividends. At the time of the 1921 ventilation study, mining had passed its peak of production and the operators were striving to reduce costs, to discover new orebodies, and to attract desirable workmen. In 1925, the camp was operating on a still less prosperous basis, but the several managements had not relaxed in their efforts to keep the mines on a paying basis, neither had they lost their faith in the ability of the mines and the price of the silver to "come back."

Early in the fall of 1921 the Bureau of Mines was invited by the mine operators at Tonopah, Nev., to make a detailed study of the underground air conditions in the Tonopah silver district. D. Harrington, chief engineer of the Safety Service Division of the Bureau, who was then directly supervising the ventilation studies, detailed district engineer B. O. Pickard, mining engineer E. D. Gardner, surgeon C. E. Kindall, and petrographer H. Insley to carry on the study. G. S. Rice, as chief mining engineer of the Bureau of Mines, was in general charge of the work. Upon the completion of the study each operator was furnished a report on conditions in his mine.

In 1925, a supplemental study at Tonopah was made by the district engineer at the request of several of the operators, and each operator advised on the changes in his respective air conditions and recommendations for improvements. Recently, several of the operators gave the

* Published by permission of the Director, Bureau of Mines.

† District engineer, U. S. Bureau of Mines.

Bureau permission to release such information as might be considered valuable in solving problems in other districts, providing the names of the individual mines under discussion were not revealed. This paper is a digest of several reports on individual mines.

CONDITIONS AFFECTING VENTILATION

Geology.—According to Spurr,¹ the country rock at Tonopah consists mainly of andesite caprock overlying earlier andesite, rhyolite, dacite, and other igneous intrusions. The main orebodies occur in the earlier rocks. The veins consist of quartz, gouge, and altered country rock.

Climate.—The difference in winter and summer temperatures at Tonopah is marked, also between night and day temperatures. The following data were furnished by the U. S. Weather Bureau at Tonopah:

Average mean temperature, 1920.....	55.4° F.
Average mean humidity, 1920.....	36 per cent.
Average mean maximum temperature.....	60° F.
Average mean minimum temperature.....	41° F.
The highest mean monthly maximum temperature (July), with a relative humidity of 75 per cent.....	86° F.
The lowest mean monthly minimum temperature (November) with a relative humidity of 17 per cent.....	24° F.
The difference between the monthly mean temperatures of July and November in 1921 was.....	45° F.

Mine Developments.—Seven companies are operating in the district with over 20 shafts. The collar elevations vary from 5900 to 6370 ft. above the sea level, most of them being between 6000 and 6200 ft. The deepest shaft is 2400 ft. below its collar, bottomed at 3550 ft. above sea level. With but two or three exceptions, none of the other shafts has a greater depth than 1600 ft. below its collar, or less than 3000 ft. above sea level. There are several stope openings and many raise openings to the surface. The properties considered in this report are all connected underground and often at several different horizons. The underground developments extend from north to south about 3 m. and from east to west approximately 2 m., making the aggregate developments of the different horizons on the several veins exceptionally large for a small district.

Mining Methods.—The orebodies vary in shape and size, in direction of strike and angle of dip, not only with each mine but at different places in each mine, which result in many different methods of mining. In 1921, one company was robbing pillars and undercutting and caving a large low-grade "horse." Another was mining an orebody by shrinkage stoping, supplementing the method by using stulls as the ore was drawn, to prevent dilution with waste from the walls. Another used square sets in the stopes, or else stulls. In a fourth mine, drifts and cross-cuts were run

¹ J. E. Spurr: *Geology of the Tonopah Mining District*. U. S. Geol. Survey *Prof. Paper* 42 (1905), 295.

underneath the flat orebody and raises put up to the ore; the broken ore was shoveled into or trammed in wheelbarrows to chutes. In another, ordinary underhand stoping methods prevailed. Stope drilling practices varied in 1921; both wet and dry drilling was practiced in each of the mines. Water for wet drilling was furnished, but many of the miners strenuously objected to its use. Blasting practices and blasting periods were not standardized or coordinated in the district, or for that matter in the individual mines. Reinforced by the recommendations of the bureau, all dry-drilling practices were stopped after the 1921 study. Blasting hours were better regulated, and attempts were made to allay the dust produced by mining.

DISCUSSION OF VENTILATION STUDY, 1921

Methods Used in the Study.—The total volume of air entering and leaving the mine was measured and the air currents were traced through the live workings. Air velocities were measured with an anemometer; barometric pressures were obtained with an aneroid barometer. Many temperature and humidity observations were made throughout the mine workings, using a Bureau of Mines sling psychrometer. Air samples were taken from nearly all working places, in order to obtain the normal condition of the working places; also, samples were taken in places in the several mines where abnormal conditions were suspected. Special vacuum tubes holding about 250 cm. were used to take these samples; the tips were broken off and the tubes were filled with the atmosphere being sampled, then carefully sealed with wax and forwarded to the Bureau of Mines laboratory at Pittsburgh for analysis.

General Conditions.—In 1921, each mine depended entirely on natural ventilation and compressed air for delivery of air to its working places, which generally proved to be expensive and ineffective. After 1921, the managements, without exception, gave more thought and attention to their underground air problems and each made some attempt to better underground working conditions. The general conditions existing throughout the district in 1921 are given for comparison with those of 1925.

The fact that all mines were inter-connected justifies the statement that uncontrolled air was permitted to circulate at will, and more often than not the air used in working places was intaking from some other mine's workings. Most of the intake air to these mines entered through one large mine, whose surface openings were extensive and generally at a lower elevation than those of the other mines, although a few surface openings at the latter normally had a slight intake current.

The main hoisting shafts of each of the larger mines which were also used as manways, however, were return airways during most of the year. One of the large mines, and several of the smaller ones, depend entirely

upon other mines for their natural ventilation currents. Ventilation in dead ends and usually in stopes was supplemented by compressed air escaping from the drills or valves in compressed air lines.

DETAILS OF UNDERGROUND ATMOSPHERES

Air Volumes.—Air volumes in three of the mines were as follows in 1921:

	Cu. Ft. PER MIN.
Mine A—Total natural air intaking at time of the study.....	26,100
Compressed air.....	4,000
Total.....	30,100
Mine B—Total natural air intaking at time of the study.....	37,300
Compressed air.....	2,000
Total.....	39,300
Mine C—Total natural air intaking at time of the study.....	31,300
Compressed air.....	2,000
	33,300

However, as is prevalent in mines dependent on natural ventilation, a very small proportion of the fresh air circulating in the mine found its way to the main working places.

Velocities.—In mine A the highest *velocity* on any level having an ordinary cross-section (24 sq. ft.) was 460 ft. per min., and in mine C, 220 ft. per min. Velocity in the working places as a whole was so slight that it was not measurable. Short circuits of mine air-currents were observed in each property.

Temperatures (1921)

From the 1000 or more temperature readings, the following were selected at random at different places in each of the several mines from intake and return air currents and in dead ends:

STATIONS, DRIFTS AND CROSSCUTS ABOVE THE 600 LEVEL			STATIONS, DRIFTS AND CROSSCUTS BETWEEN THE 600 AND 1000 LEVELS		
WET BULB, °F.	DRY BULB, °F.	RELATIVE HUMIDITY PER CENT.	WET BULB, °F.	DRY BULB, °F.	RELATIVE HUMIDITY, PER CENT.
40	50	38	49	54	70
56	58	89	54	60	68
52	59	62	49	60	47
65	69	82	70	72	91
68	70	91	72	72.5	98
72	73	96	69	73	83
68	76	68	80	80	100
			80	81	96
			78	81	92
			77	84	74
			73	86	55
			85	88	89
			83	89	78

STATIONS, DRIFTS AND CROSSCUTS BETWEEN THE 1000 AND 1500 LEVELS

WET BULB, °F.	DRY BULB, °F.	RELATIVE HUMIDITY, PER CENT.
53	53	100
54	56	89
53	60	63
75	75	100
75	79.5	82
79	80	96
80	80	100
74	81	76
70	81	67
62	83	32
83	83	100
84	84	100
80	85	81
85	88	89
97	101	87
98	102	87
88	103	56
114	114	100

STOPE TEMPERATURES

No air movement except compressed air—exception is mine 4

	WET BULB, °F.	DRY BULB, °F.	RELATIVE HUMIDITY, PER CENT.
Mine 1			
Average.....	80	85	82
Minimum.....	71	78	72
Maximum.....	78	88	65
Maximum humidity.....	85	86	96
Mine 2			
Average.....	72	74	91
Minimum.....	69	70	95
Maximum.....	77	79	90
Maximum humidity.....	72	73	97
Mine 3			
Average.....	74	78	83
Minimum.....	48	56	58
Maximum.....	90	91	96
Maximum humidity.....	80	80	100
Mine 4			
Average.....	58	63	78
Minimum.....	48	55	66
Maximum.....	71	76	77
Maximum humidity.....	60	61	95
Mine 5			
Minimum stope.....	78	91	57
Maximum stope.....	98	102	87

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WATER AND ROCK TEMPERATURES, °F.

Mine A	
1200-ft. level (rock).....	102
1400-ft. level (water).....	110
1400-ft. level (water from bore hole).....	114
Mine B	
Surface water.....	55
1100-ft. level at face of new development (rock).....	87
1400-ft. level (water).....	88
1500-ft. level (water).....	94
Mine C	
500-ft. level (rock).....	76
800-ft. level (rock).....	86
960-ft. level (rock).....	87
Mine D	
1200-ft. level (rock).....	80
1600-ft. level (rock).....	91
1800-ft. level (rock).....	94, 98, 100
1600-ft. stope (rock).....	86
1600-ft. No. 2 stope (rock).....	103
1700-ft. stope (rock).....	97

MAXIMUM TEMPERATURES

LEVEL	WET BULB, °F.	DRY BULB, °F.	HUMIDITY, PER CENT.
1400-ft. level.....	114	114	100

(Dead end—no circulation)

Gas Analyses

Many samples of the air were taken in the working places, to ascertain the quality of air which was being breathed by the miners. The following résumé will give a general idea of conditions in the district in 1921:

COMPOSITION OF MINE-AIR SAMPLES FROM 6 MINES OF TONOPAH DISTRICT IN 1921^a

MINE	GAS	IN LEVELS.	IN STOPES.
		PER CENT. BY VOLUME	PER CENT. BY VOLUME
No. 1	CO ₂	0.08 to 0.15	0.05 to 0.28
	O ₂	20.94 to 20.62	20.88 to 20.67
	N ₂	78.96 to 79.96	79.02 to 79.11
No. 2	CO ₂	0.04 to 0.18	0.05 to 2.91
	O ₂	20.93 to 20.57	20.91 to 17.08
	N ₂	79.03 to 79.31	78.96 to 80.01
No. 3	CO ₂	0.05 to 0.24	0.04 to 0.62
	O ₂	20.95 to 20.61	20.92 to 20.14
	N ₂	79.00 to 79.41	79.02 to 79.57
No. 4	CO ₂	0.10 to 0.13	0.09 to 5.03
	O ₂	20.88 to 20.36	20.87 to 20.56
	N ₂	79.02 to 79.15	79.02 to 79.22
No. 5	CO ₂	0.05 to 1.77	
	O ₂	20.93 to 14.69	
	N ₂	79.02 to 83.63	
No. 6	CO ₂	0.05 to 0.37	
	O ₂	20.87 to 20.56	
	N ₂	79.67 to 79.09	

^a Percentage in normal air; by volume: CO₂, 0.03; O₂, 20.93; N₂, 79.04.

The Bureau of Mines field men believe from experience that in working places where practically no movement of air exists, with the temperature over 70° wet bulb and humidity near the saturation point, the air supply theoretically should be maintained as pure as surface air, and when the carbon dioxide content exceeds $\frac{1}{4}$ of 1 per cent. (0.0025) or the oxygen content is less than 20 per cent., the efficiency of workmen is lessened and their health impaired. Seldom did the working-place atmospheres sampled show enough oxygen deficiency or excess of carbon dioxide to affect the workmen visibly, but on the other hand the air was subnormal in nearly every sample and showed the results of short-circuiting, mixing or stagnating. No appreciable amounts of carbon monoxide were observed in any of the working places.

Rock-strata Gases

A number of dead-end raises and some abandoned workings in the district contained atmospheres in which a carbide lamp would not burn. The raises are in widely separated localities and in different rock formations. One of two raises in the same formation may contain a non-inflammable gas and the second may be free from it. However, the non-inflammable gas was discovered in nearly every mine in one or more places. Men have been overcome and sometimes have lost their lives when unexpectedly opening up a pocket of this gas in new developments. Analyses by the Bureau of Mines showed the gas to be principally nitrogen with some carbon dioxide.²

In one mine a raise was sampled that was full of the gas with a small amount flowing out in the drift; a sharp line of demarcation occurred at the bottom of the raise above which a carbide light was extinguished. This raise had been driven on a 1-ft. quartz vein in silicified rhyolite. Samples of the gas analyzed as follows: In the drift beneath the raise at the point where the light was extinguished, CO₂, 0.08 per cent.; O₂, 20.7 per cent.; N₂, 79.22 per cent.; 15 ft. above the bottom of the raise, CO₂, 0.62 per cent., O₂, 11.54 per cent., N₂, 87.84 per cent.; 25 ft. above the bottom of the raise, CO₂, 0.90 per cent.; O₂, 5.89 per cent.; N₂, 93.21 per cent. Samples, taken at another isolated occurrence at the same mine, analyzed: CO₂, 0.38 per cent., O₂, 14.90 per cent.; N₂, 84.72 per cent. In another mine, gas analyses showed CO₂, 1.26 per cent.; O₂, 11.87 per cent.; N₂, 86.87 per cent.

It was noticed that where any dead-end workings near such gas emanations had been abandoned for a few months, the atmosphere showed a deficiency of oxygen and a relatively high percentage of nitrogen and carbon dioxide.

² E. D. Gardner: *Rock Strata Gases of a Nevada Mining District. Serial 2427, Bureau of Mines (1922) 2.*

Dust in Mine Air

Dust in metal mines probably causes more sickness and ultimately more deaths among the workers than is generally appreciated even by the miners themselves. The effect of continued inhalations of exceedingly minute sharp and insoluble rock particles is well known, but strangely enough "getting the dust" is risked by some miners in the hope that they will be the exception. It is probably another illustration of the old belief "that what we cannot see will not hurt us."

Allaying of dust and dust control are relatively simple and inexpensive processes and are effective, provided that strong air currents are furnished to remove the dust from the workings. There are still some metal-mine operators who are prone to ignore the hazard and hesitate to incur the *expense* (?) of positive strong air currents. But dust, poor air and high temperatures in working places have done more to drive good miners to other industries than any other hazard underground, and operators throughout the metal-mining district are paying the penalty of procrastination by being forced to operate with relatively incompetent workmen.

Causes.—Dust in mine atmospheres is made by drilling, blasting, shoveling, drawing chutes and shrinkage stopes, by the stirring up of settled dust by air currents, drill exhausts or compressed air blowers, and by other mine operations.

Tonopah Conditions in 1921.—As will be noted by the relative humidity in the tables, the mines in the Tonopah district are comparatively dry. The ore contains but little included moisture and, in common with most metal mines, considerable dust is made by all mining operations except where water is freely used. Some dust enters the air every time the dry material is handled and the longer the ore stays in the mine the dustier it becomes. The mine operators at Tonopah, although they did not have accurate data on their dust conditions, realized the hazard to a certain extent and provided water lines throughout the mines to supply water to allay the dust; often sprays have been installed at the top of ore chutes. Dry drilling underground or handling of broken ore before it was sprinkled was prohibited by mine order, but these rules were not always complied with and in some isolated instances the furnishing of water connections to certain working places had been delayed.

The investigators noticed considerable settled dust in various parts of the mine workings, particularly in the return airways. The concussions from blasting stirred up the dust for the workmen to breathe while going off or coming on shift. Sprinkling of airways was considered important, but it was often overlooked in favor of some more important (?) operation. The mucker was likely to forget to sprinkle his ore pile before shoveling and the trammer to neglect to turn on the spray on the chute before loading his cars, unless severely reprimanded by the shift

boss for each offense. It was necessary for the mines to carry on an educational campaign among the bosses and workmen to get them to realize the dust hazard and the possibility of eliminating the hazard through simple practices, and rules had to be enforced by discipline.

Method of Testing Dust Content.—The dust content of the air was sampled with a sugar-filter apparatus developed by the Bureau of Mines and modeled after South African equipment. A miner may be considered as inhaling 1 cu. ft. of air per min. while at maximum effort; the mine air, therefore, was aspirated by means of a bicycle pump through the sugar at this rate, and the sugar tubes were held as close as was practicable to the workmen's heads, so that the sample would indicate the amount and kind of dust that they were breathing.

In order to compare this work with that done in South Africa, the samples were taken in periods of 15 min. each. The sugar tubes containing the dust sample, after being treated with formaldehyde, were mailed to the Pittsburgh Station of the Bureau of Mines to be analyzed, measured, and photographed.

Conclusions.—More than 100 dust samples were collected in the district, and physical examinations, supplemented by chest X-rays, were made of many workmen of the district. Although details of this phase of the study are not in order in this paper, it was generally concluded that the dust condition in the Tonopah district warranted immediate control if the health hazard was to be considered, and extensive recommendations offered to assist the operators.

VENTILATION IN 1925

As a result of the detailed data which were furnished the Tonopah operators in 1921, two of the companies determined upon extensive ventilation plans. The 1925 study revealed that one company had installed a pressure fan on the surface, dedicating one shaft entirely as an air intake; had installed many doors and bulkheads and had intelligently distributed fresh moving air to many of its working places.

A second company placed two medium-sized fans in the bottom of its mine and, by the extensive use of brattice cloth for curtains and for bulkheads to isolate certain dead workings, supplemented by many standardized, well built, wooden doors, had provided a well planned intake airway. It was noted, however, that at each mine the main hoisting shafts had been retained as return airways.

In other properties, small blowers with tubes were ventilating certain workings, or rather were providing air movement where heat conditions were excessive. Several mines operating in 1925 had given serious attention to dust-allaying processes and had tried to standardize blasting practices and to regulate blasting hours.

FEATURES OBSERVED IN 1925

Surface Fan at One Mine Only

The only surface fan in the Tonopah district is a blowing reversible fan with a computed capacity of 50,000 cu. ft. per min. when run at 3.75 in. water gage, 42 b. hp., elevation 6000 ft. It is belt-driven by a 50-hp., 690-r. p. m. motor. The entire installation is housed in a fireproof steel structure. The total cost of the surface installation, including freight (excluding the motor which was transferred from another job) was \$8100. The fan installation made this particular mine entirely independent of all the other mines in the district for its intake air, and, except for a little air delivered to an adjoining mine, all of the returned air was delivered directly to the surface through workings belonging to the property. The levels were well supplied with wooden doors, with latches, generally in wooden frames; a few of the doors were in pairs.

Fans and Pipes Underground

The same mine was using small fans with pipes for ventilating dead-end developments. Many different types of fans were used underground, but the ventilation engineer planned to experiment to determine the most satisfactory unit before buying more. The company manufactured all of its galvanized tubing and had standardized on 12-in. diameter pipe for short dead ends and 16-in. pipe for lengths in excess of 1000 ft. When two or more faces were ventilated by one fan, 16-in. pipe was used to the junction and a 12-in. pipe from there to the development faces. Very little canvas or other flexible tubing was being used at the time of this study, although it was being considered for short lengths at the face.

Intake Air from Other Mines

At a second mine the main ventilation power-unit was established on the bottom level in the vicinity of the mine shaft. Two fans, both No. 7 Sturtevant, one with a 25-hp. motor, the other with a 60-hp. motor, were pulling a total of 15,000 cu. ft. per min. of air through a 4 by 6 ft. cross-cut, which was used as a common intake. The air was forced by the fans through galvanized pipes to three important, hot, development faces on the bottom level, from which faces the air was permitted to return through the main shaft to the surface. The temperature of the intaking air in the crosscut was 70° wet bulb and 80° dry bulb, humidity of 61 per cent.

A well-planned intake airway had been provided using brattice cloth for curtains, bulkheads, and overcasts, supplemented with wooden doors with latches. However, the larger part of this intake air was being supplied for other properties through several inter-connections. A small part was fresh air from the surface through what was known as the intake shaft.

A third company adopted the use of small underground blowers to supplement the natural air currents which were intaking from other mines. The blowers distributed the air through pipes to dead ends.

Improvement in Conditions

The principal changes through mechanical ventilation in the first mine were in quality of air in the working places near the main ventilation current and the general lowering of temperature of from 1° to 10° in the intakes. Where fresh air was being used, the carbon dioxide content did not exceed 0.05 per cent. in any instance, but after the air had passed through old stopes and other workings the carbon dioxide content increased to 0.14 per cent. Stope temperatures were not much affected, as no definite provision had been made for delivery or circulation of air in stopes, and as a result, miners were resorting to compressed air "blowers."

The temperature of the air delivered by the main surface fan at this mine varied in the several levels as follows:

LEVEL	WET BULB, °F.	DRY BULB, °F.	HUMIDITY, PER CENT.
1500-ft.	54	69	36
1600-ft.	61	70	59
1800-ft.	64	74	58
2000-ft.	66	76	59
2100-ft.	69	78	64
2200-ft.	78	81	87
2400-ft.	79	83	92

The rapid increase in humidity below the 2100 level is accounted for by a relatively small quantity of air and an increased amount of exposed water in the shaft.

The return air in the main shaft had an average temperature of 80° wet bulb, 83° dry bulb, and 80 per cent. humidity, which averages, 8° less in dry bulb and 2 per cent. less in humidity than in 1921, although much water had been developed in the lower levels of the mine since then.

The maximum total volume of air furnished this mine in 1921 by natural ventilation was 31,000 c. f. m., whereas by mechanical ventilation 55,000 c. f. m. were furnished, an increase of 75 per cent. over maximum natural conditions, and probably several times over average natural conditions.

On one of the main haulage levels in this mine, very satisfactory operation of mechanical haulage doors of a patented commercial type was noted. By estimate, the doors were not opened more than 20 times per shift and, as they were closed quickly and automatically, the effect on ventilation was not noticeable on the other levels. Their importance as a ventilation unit is shown by the fact that when open, nearly one-half

of the total mine air was short-circuited to the main shaft, whereas when closed, only 3000 c. f. m. leaked through to the ore-transfer station at the main hoisting shaft.

VENTILATION ORGANIZATION

The company operating the surface pressure-fan gave their chief engineer charge of the ventilation underground in addition to his other duties. He was able, however, to obtain more or less cooperation from the ground bosses, except when production problems took precedence. The company that installed the intake airway, through abundant use of brattice cloth, had one young engineer whose sole duty was the installation of doors, bulkheads, curtains and like devices, and the placing of auxiliary fans and tubing for air distribution.

The Bureau of Mines stresses the importance of a complete organization for the ventilation work underground, headed by an interested engineer. Mine conditions are continually changing, and distribution of air must keep up with the new working places, just as must car tracks and compressed-air lines, if efficiency is the watchword. Ventilation can not be left to the individual production and development bosses, for it is impossible for them to coordinate the proper distribution of air currents throughout the whole mine. Ventilation is a problem of the entire mine; the individual working places are only a step in the solution.

CONCLUSIONS

The following discussion is extracted from the several individual reports prepared by the author in 1922.

Although ventilation is very important in the successful and efficient operation of a mine, it must be considered (1) subsidiary to the main problem of production, just as is stoping or pumping and (2) dependent on the plans and policy of the management.

In ventilating any mine, one must first consider where and why fresh moving air is needed, the quantity required to bring the conditions in the working place up to the maximum requirements for comfort, and then consider the best available main intake and return air courses, the location of the surface fan, if one is required, and finally the most effective distribution of air.

The Tonopah District

Much time was spent in developing satisfactory ventilating schemes for the Tonopah mines, and it was concluded, after an analysis of the data, that the only satisfactory method for conquering the ventilation and fire-hazard problems of the several mines was to install a large fan either on the surface or underground at each property, and to control the

air currents in direction and volume by doors and regulators with supplementary distribution in dead ends and stopes by small blowers and pipes.

In making recommendations for efficient ventilation, the following major points were taken into consideration:

1. The size and life of each mine and status as a profitable commercial enterprise.
2. Places at present and in future which must be ventilated, that is, supplied with fresh moving air.
3. Type of ventilation system—pressure or exhaust, or both.
4. Intake air courses:
 - a. Cross section of openings.
 - b. Shortest route, that is, distances.
 - c. Prevention of contamination by heat, spores, dust, and by powder, rock, and timber gases.
 - d. Relation of rock and surface temperatures.
 - e. Course and direction of natural air currents.
 - f. Interferences with tramming and hoisting.
 - g. Elimination of inter-ventilation between mines under separate managements.
5. Return air courses:
 - a. Cross section of openings.
 - b. Condition of old workings.
 - c. New openings required.
 - d. Best routing to secure minimum of timber decay.
6. Proper location of main fan and the distribution of air through proper location of doors, splits and auxiliary fan.
7. Size of main fan:
 - a. The amount of air (c. f. m.).
 - b. Mine resistance (pressure of water gage).
 - c. Size of motor.
8. Fire and dust hazards.
9. Cost.
10. Organization for installation, operation and maintenance.

It was believed that at least four of the properties were large enough and had a future as a profitable commercial enterprise sufficiently promising to warrant serious consideration of the ventilation problem and its solution through the positive control of air currents by mechanical means.

The several connecting mines, with scattered and often isolated units in each requiring individual ventilation, and the many restricted cross-sections of possible air courses offer unusual and difficult ventilation problems which can only be solved by a careful study of conditions and an accurate knowledge of each property.

In properties of this nature, the total and relative air volumes should be based on the number and the size of the workings, the rock temperatures, the distances from the intake and other conditions, rather than quantity per individual worker. The chief obstacles in these mines are dust, inert gas, and the high temperatures and relative humidities in the workings, which can only be neutralized by fresh moving air at a minimum velocity of 100 ft. per minute.

According to Harrington,³ the Bureau of Mines ventilation standard is as follows:

Efficient ventilation of metal mines consists in supplying at all times such volumes of circulating pure air at all workings as will enable the miner to work in comfort at maximum physical capacity without endangering his health.

Metal mines dependent on natural ventilation fail to meet the standard requirements of delivery and distribution, as above stated, in that normally there is very little air entering the mine; the air currents are extremely variable; a considerable percentage of the fresh air is returned to the surface without being used in the working places; much good air is diluted with used air; few of the hot working places get the benefit of fresh moving air.

Airways should always be kept free of obstructions and fire hazards.

Other Districts

Many publications on ventilation are available, some by the U. S. Bureau of Mines, that cover conditions in other districts, and point out standard and ideal practices and stress the dangers of sub-standard conditions. To these publications the writer refers all engineers who are interested in the general subject and who desire to ascertain wherein further lessons can be learned from the air conditions and the ventilation operations at Tonopah.

ACKNOWLEDGMENTS

The author hereby acknowledges that the 1921 conditions were freely extracted from unpublished reports by Gardner, Kindall, and Insley. He is also grateful to the mine operators at Tonopah who so generously cooperated in the studies.

DISCUSSION

W. S. WEEKS, Berkeley, Calif. (written discussion).—The author presents some interesting statistics on underground temperatures in the Tonopah district. It is noteworthy that in most of the working places the wet-bulb temperature was such that they could be made comfortable by moderate air velocities. An important point in ventilation work is emphasized in the paper as follows: "The Bureau of Mines stresses the importance of complete organization for ventilation work underground, headed by an interested engineer."

Ventilation has progressed to the point that it is now on a fairly sound mathematical basis. It is an economic crime to attempt to solve by costly experiments problems that can be attacked successfully by the use of mathematics and mechanics. It is just as unreasonable to attempt to design a ventilating system without a knowledge of the physical principles involved as it is to attempt to decide on the contents of a complex furnace charge without a knowledge of chemistry. It is

³ D. Harrington: Ventilation in Metal Mines (A Preliminary report). *Tech. Paper 251*, Bureau of Mines (1921) 35.

possible to determine the correct mixture of ores and fluxes by a series of expensive trials. Such methods are obsolete in the smelter, but the miner seems loathe to take advantage of the knowledge of ventilation that is now at hand.

It is only by having at least one man who understands the principles of ventilation that the operator can get his money's worth out of the investment in equipment. In case of fire, this man is absolutely essential, for a few minutes' delay in the proper handling of the ventilating currents may mean the sacrifice of men. I was told by the representative of a fan manufacturer that at the outbreak of a disastrous fire he received a telephone call from the mine asking if the air current could be reversed by running the fan backward.

Mr. Pickard says: "The Bureau of Mines field men believe from experience that in working places where practically no movement of air exists, with the temperature over 70° wet bulb and humidity near the saturation point, the air supply theoretically should be maintained as pure as surface air, and when the carbon dioxide content exceeds 0.25 per cent. or the oxygen content is less than 20 per cent., the efficiency of workmen is lessened and their health impaired."

The experiments of Flügge and of Hill on subjects enclosed in a tight chamber with the oxygen content of the air much below 20 per cent. and the carbon dioxide much above 0.25 per cent. show that disagreeable symptoms were due not to the chemical character of the air but to the lack of cooling power. When the cooling power of the air was adequate, no disagreeable symptoms were present.

If a large man is working in a confined place he may consume as much as 3 liters per min. of oxygen. If the oxygen content of the air is to remain at 20 per cent., 10 cu. ft. of fresh air must be supplied each minute. In order to keep the carbon dioxide from rising above 0.25 per cent., 50 cu. ft. per min. of fresh air must be supplied.

A low carbon dioxide content is an indication that air is being introduced, and there is an increase of cooling power. High carbon dioxide and low cooling power usually go together and the carbon dioxide is blamed for the trouble caused by heat stagnation

Ventilation of the Liberty Tunnels at Pittsburgh

By LOUIS W. HUBER,* PITTSBURGH, PA.

(New York Meeting, February, 1927)

THE Liberty tunnels extend through a very steep hill in Pittsburgh (locally called Mount Washington) for a distance of slightly over a mile. The two tunnels parallel each other and are 59 ft. apart, center to center (Fig. 1). They are for vehicular traffic, and in each there is ample space for two lines of vehicles. Northbound traffic, toward downtown Pitts-



FIG. 1.—NORTH PORTAL OF LIBERTY TUNNELS. NOTE THE WINDBREAK AT END OF EAST TUNNEL.

burgh, is confined to the east tunnel and southbound traffic to the west tunnel.

The South Hills residential district is separated from downtown Pittsburgh by Mount Washington, a steeply rising escarpment south of and adjacent to the Monongahela River. The Mount Washington district immediately south of the Monongahela River has been reached, for many

* Mining Engineering Dept., Carnegie Institute of Technology.

years, by means of inclined planes for both passengers and freight, but the districts beyond Mount Washington could be reached only by following long circuitous routes against excessive grades. Finally, after years of careful planning and deliberation, the Liberty tunnels were built.

Work on the tunnels was begun in December, 1919, and they were virtually finished in May, 1924. They were opened to traffic at that time, but the ventilating system was not put in operation until August of that year.

As finally constructed, the Liberty tunnels consist of two tubes 59 ft. apart on centers and 5889 ft. long from portal to portal. This length includes a 66-ft. air chamber at the exit end of each tunnel. These air chambers act as windbreaks. The grade of the tunnels is 0.329 per cent. descending toward downtown Pittsburgh. The springing line of each tunnel is 7 ft. 6 in. above the top of the roadway and the diameter of the springing line is 26 ft. 6.5 in. The roadways are 21 ft., between curbs 1 ft. high; there is a 4-ft. sidewalk on the right of the roadway in each tunnel, which is protected by a heavy pipe handrail.

CARBON MONOXIDE CONTENT OF TUNNEL AIR

At the time of their completion, the Liberty tunnels were the longest ever built for vehicular traffic. From tests made by the U. S. Bureau of Mines for the New York-New Jersey Interstate Tunnel Commission,¹ it was concluded that, in mixed traffic (trucks and passenger cars), the average amount of carbon monoxide gas given off in the exhaust from the vehicles could be assumed to be 1.5 cu. ft. per min. From actual physiological tests, it was determined that 4 parts of carbon monoxide in 10,000 parts of air is the maximum concentration to which a human being can be continuously exposed for 1 hr. without noticeable effect; or that, if the carbon monoxide concentration is increased uniformly from zero, the maximum concentration without noticeable effect is 6 parts per 10,000, or an average of 3 parts per 10,000 over the 1-hr. period.² Since the proposed ventilating system would have permitted a uniformly rising concentration from zero to a maximum in the direction traffic was traveling, a maximum concentration of 6 parts of carbon monoxide per 10,000 parts of air was selected. The fresh-air requirement for each automobile under these conditions is 2500 cu. ft. per min., or enough to dilute the 1.5 cu. ft. of carbon monoxide assumed to be given off per automobile per minute to the allowable maximum concentration of 6 parts in 10,000. The maximum number of cars in each tube under full

¹ See report of the New York State Bridge and Tunnel Commission, 1921, pp. 91-140.

² Henderson, Haggard, Teague, Prince, and Wunderlich: Physiological Effects of Automobile Exhaust Gas and Standards of Ventilation for Brief Exposures. *Jnl. Indust. Hygiene* (August, 1921) 72, 137.

normal traffic conditions was assumed to be 113; this number presupposes a double line of motor vehicles in each tube with the cars spaced 100 ft. apart. The total fresh-air supply for traffic on this basis is 280,000 cu. ft. per min. for each tube.

VENTILATION SYSTEM

The ventilation system (Figs. 2 and 3) now in operation provides for an equal division of duty between the exhaust and supply fans on each tube. There are eight fans in all, four fans operating on each tube; two as upcasting, or exhaust, fans, each normally handling 70,000 cu. ft. per min., and two as downcasting, or supply, fans, each of which also normally handles 70,000 cu. ft. per min.

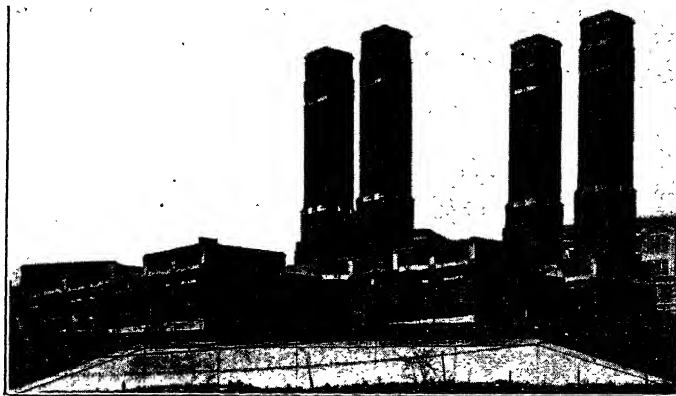


FIG. 2.—POWER PLANT OF LIBERTY TUNNELS. THE FOUR STACKS ARE FOR EXHAUST AIR.

The tubes were opened to traffic before the ventilation system was in operation, so that for a period of about four months they were ventilated only by means of whatever natural ventilation prevailed in addition to the ventilation induced by the lines of traffic moving through them. During this period determinations were made by Bureau of Mines engineers to determine the ventilation effected by these agencies.³

The tunnels are parallel twin tubes, with the same difference in elevation (19 ft.) between the portals, therefore the natural ventilation effect should be the same in each, discounting the effect of the wind baffles at the north end of the east tunnel and at the south end of the west tunnel. Readings taken while there was no traffic in either tube showed that velocities from 0 to 400 ft. per min. on the average were set up by natural ventilation, and in exceptional cases velocities of 700 to 800 ft. per min.

³ A. C. Fieldner, W. P. Yant, and L. L. Satler, Jr.: Natural Ventilation in the Liberty Tunnels. *Eng. News-Record* (1924) 93, 290-291.

sufficient for several times the number of cars that caused it. During this period, on several occasions, the traffic was at the rate of 900 cars per hr. and the induced velocity under these conditions was 600 to 800 ft. per min. in the tube in which the traffic opposed a natural velocity of about 200 ft. and was 800 to 1000 in the other tube. A careful carbon monoxide content check was kept during this period, and, in but a very few instances, when a particularly strong wind opposed the induced ventilation in one of the tubes, was it found necessary to stop traffic to allow the air current set up by natural means to clear out the gas. The maximum carbon monoxide content was 2 parts in 10,000; it was less than 2 parts almost all of the time and usually was less than one part in 10,000.

When the tubes were started, it was intended to ventilate them by a continuous current of air from one end to the other. The original intention was to introduce air at high velocity a short distance from the tunnel mouth, at a slight angle with the longitudinal axis, through an opening in the periphery into the tunnel and toward the opposite end. It was believed that this air in addition to that drawn in and forced through by the injector effect would be sufficient for safe ventilation. Construction was not under way very long, however, before it was foreseen that, while this system of ventilation might be adequate for normal conditions, the air velocity at the inlet nozzle when providing for excess capacity would be excessive. It was decided to locate the ventilating plant on the hill-top near the midpoint of the tunnels rather than at either end.

VENTILATING SHAFTS

Each tube has its own separate ventilating shaft which is divided into four compartments, two for exhaust air and two for fresh supply air. The compartments are 10 ft. 10 in. by 13 ft. each; the supply compartments are each 5 ft. $11\frac{1}{4}$ in. by 13 ft. The shafts are both 200 ft. deep; this is measured from the floor of the fan house or power house to the roofs of the tubes. The shaft for the west (southbound) tunnel is 39 ft. 6 in. north of the midpoint and that of the east (northbound) tunnel 89 ft. 6 in. north of the midpoint. The shafts are not directly opposite each other, but 50 ft. apart measured along the longitudinal axes of the tubes.

This location of the power plant and air shafts removes the vitiated air after it has traveled through one-half of the tunnels, which reduces the period of contamination by one-half and lessens the frictional resistance considerably.

DIRECTION OF AIR MOVEMENT

The air movement in the Liberty tunnels is not, as is very often supposed, from both ends toward the center. The air in each tunnel moves with the traffic throughout the entire length of the tunnels, in order to

utilize the air movement set up by the moving vehicles. This ventilation is effected by drawing air with exhaust fans from the tunnel portal to the center, where it is drawn up the exhaust shaft and discharged into

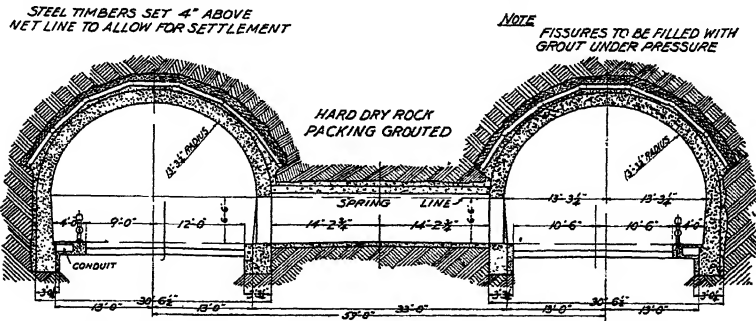


FIG. 4.—CROSS-SECTION OF TUNNELS SHOWING PASSAGEWAY BETWEEN THEM.

the atmosphere. (Fig. 5.) The air movement in the other half of the tunnel is maintained by blowing air down the supply shafts and toward the opposite portal of the tunnel through suitably designed nozzles in



FIG. 5.—INTERIOR VIEW OF TUNNEL SHOWING EXHAUST SHAFT.

the sides. (Fig. 6.) In this way half of the tunnel is ventilated by the exhaust system and half by the blowing system.

"The air movement in each tube is as follows: In the tunnel carrying traffic away from downtown Pittsburgh, fresh air enters the tunnel with the traffic at the Pittsburgh (north) portal, moves with the traffic to the

center and is there drawn out through the exhaust compartments of the vertical shaft at the same time that the fresh air is being blown through the fresh-air supply compartment of this same shaft. This fresh air moves with the traffic through the remaining half of the tunnel and is exhausted at the South Hills (south) portal.

"In the tunnel carrying traffic toward downtown Pittsburgh, fresh air enters the tunnel with the traffic at the south portal, moves with the traffic to the center and is there drawn out through the exhaust compartments of the vertical shaft at the same time that fresh air is being blown down through the fresh-air supply compartments of this same shaft. This fresh air moves with the traffic through the remaining half of the tube and is exhausted at the north portal.

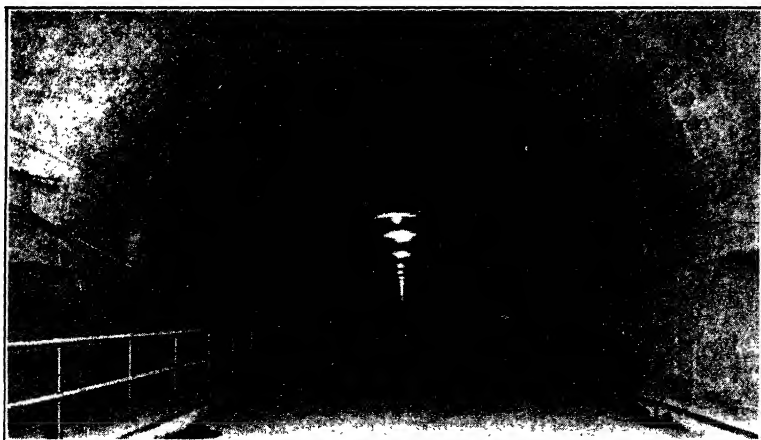


FIG. 6.—INTERIOR VIEW OF TUNNEL SHOWING INLET NOZZLES.

"Incoming fresh air from the fans is discharged near the middle of each tunnel, through suitably designed nozzles which are turned forward in the direction of traffic. This arrangement prevents short-circuiting and consequent merging of the fresh air with the vitiated air which is being drawn out through the other compartments of the same shaft. The nozzles are about 50 ft. beyond the exhaust shaft opening, and the only current of air in this space is that induced by the traffic and the incoming fresh air."⁴

EXHAUST AND BLOWING FANS

It was planned that the duty of supplying the required 280,000 cu. ft. of fresh air per minute to each tube should be equally divided so that

⁴ C. K. Harvey: The Liberty Tunnels and Power Plant. *Proc. Engineers Society of Western Pennsylvania* (1926) 42, 255-283.

each exhaust fan would draw 70,000 cu. ft. per min. from its tunnel and each supply fan blow 70,000 cu. ft. per min. into its tunnel. On this basis, the volume load on each of the four fans on each tube would be 70,000 cu. ft. per min. However, it was decided to furnish excess capacity, so the volume duty of each fan was set at 140,000 cu. ft. per min. This provides 100 per cent. overload capacity over that required for normal full traffic conditions in the tunnels. The pressure required at each of the blowing fans for the volume of 140,000 cu. ft. per min. is 1.7 in. of water and that required at each of the exhaust fans, 0.62 in. of water.

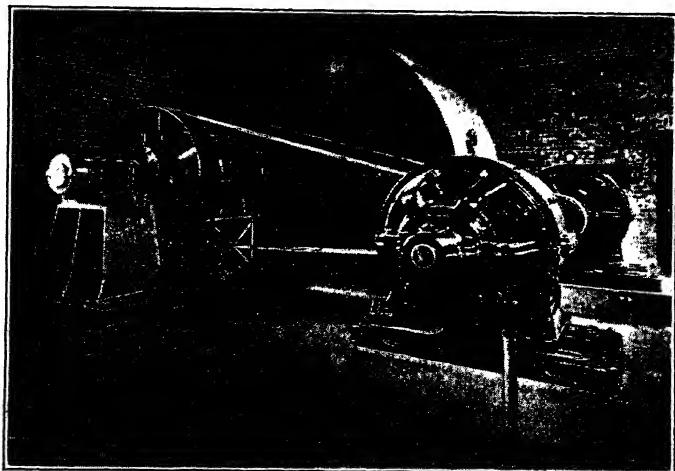


FIG. 7.—VIEW OF FAN UNIT SHOWING TWO-MOTOR DRIVE.

The fans are the familiar forward-curved multiblade wheel type. The wheels are 115 in. diameter and 54 in. wide. The exhaust fans are driven at 90 r.p.m. and the supply fans at 140 r.p.m. to deliver 140,000 cu. ft. per min. The fans are belt connected to variable-speed direct-current motors; 40-hp. motors are furnished for the exhaust fans and 80-hp. motors for the supply fans. One motor per fan is sufficient to drive them at their rated speeds, but to provide for 25 per cent. excess capacity to take care of blockades and other abnormal traffic conditions each fan is equipped with two motors. These are arranged so that the 100 per cent. overload capacity speed can be increased 25 per cent., and 25 per cent. more air can be delivered to the tunnels. The motors are belt connected to the fan shaft (Fig. 7). Magnetic clutches are provided on the fan-shaft pulleys so that either or both motors can be used for driving the fans. The clutches are interlocked and controlled in such a way,

however, that it is impossible to engage the pulley clutch of the spare motor until each motor is up to 100 per cent. excess capacity speed.

The capacity of each fan for normal full traffic conditions is 70,000 cu. ft. per min. With either of the two motors, this can be increased to 140,000 cu. ft. per min.; with both motors the volume delivered can be increased to 175,000 cu. ft. per min. without in any way overtaxing any of the equipment. It is apparent, then, that two and one-half times as much air as is required under normal full traffic conditions can be furnished to the tunnels. It is believed that this equipment will provide adequate ventilation for any contingency that can be foreseen.

Because the South Hills high school is very near the fan and power plant, the exhaust fans do not discharge directly into the atmosphere. Brick stacks 110 ft. high are provided for the exhaust, so that it may be discharged at a greater height than that of the school building. There are four stacks, each 10 by 10 by 110 ft. high.

CARBON MONOXIDE RECORDER

A continuous carbon monoxide recorder developed at the Pittsburgh Experiment Station of the U. S. Bureau of Mines rendered invaluable service when the tunnels and their ventilating plant were put in operation. This ingenious device furnishes a graphic record of the carbon monoxide concentration in the tunnels over a 24-hr. period and is sensitive to one to two parts per million. Since the speed at which the ventilating fans operate is dependent on the carbon monoxide concentration in the tunnels, the value of such a recording device is apparent. When the carbon monoxide concentration approaches 4 parts per 10,000 the fan output is increased and when the concentration becomes less the fan speeds are reduced. This results in a great saving in power, in less supervision, and in protection against false claims of injuries suffered through bad air conditions in the tunnels.

The present practice is to keep one supply and one exhaust fan in operation on each tube throughout the twenty-four hours and to use all fans during the day only. The speeds of all fans are governed by traffic conditions, which are regular and well known, so that fan speeds are increased and decreased at definite times during the day to provide for traffic peaks and depressions in advance of their occurrence.

EMERGENCY CROSS TUNNELS

At 500-ft. intervals starting from the north portal of the tunnels, there are 11 cross tunnels, 6 ft. 6 in. by 4 ft. wide and 33 ft. long, from one tunnel to the other. (Figs. 3, 4.) These are closed at each end with steel doors, which are kept locked; the doors provided with small glass windows

to be broken in case of emergencies, so that the doors may be unlocked. The cross tunnels are intended to provide means of exit from one tunnel into the other in case of fire, traffic blockade, or other emergencies.

At the exit end of each tunnel the sides are extended for a distance of 75 ft. beyond the hillside. This portion is left uncovered and is designed to act as a windbreak. The principle of operation is that a head-on wind toward the exit end of the tunnel will meet the outcoming air and the resultant air travel will be upward through the uncovered portion of the tunnel. This windbreak seems to be very effective.

The design and construction of the Liberty tunnels was carried on under the supervision of A. D. Neeld, consulting engineer of Pittsburgh. Stanley L. Roush, architect, of Pittsburgh, designed and supervised all architectural features, and Booth and Flinn, Ltd., of Pittsburgh, were the contractors.

The tunnels are operated under the jurisdiction of the Department of Public Works of Allegheny County, Pennsylvania. The force required to operate the tunnels and power plant consists of 20 men: one superintendent, one clerk, three plant operators, four assistant plant operators, two electricians, five utility men, two traffic policemen and two watchmen. In 1925, \$68,981.50 was appropriated for the operation of the tunnels, of which \$16,500 was spent for electric current and \$38,677 for salaries and wages.

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DISCUSSION

O. SINGSTAD, New York, N. Y.—One feature of the ventilation method used in Pittsburgh is very interesting; that is, the division of the longitudinal ventilation system into two units. I believe that method has never before been put into practice. There were many engineers, including myself, who feared that a large part of the vitiated air would pass by the uptake shaft and into the second section. Mr. Huber's statement that this feature works satisfactorily is reassuring but further details would be desirable.

About the time the people in Pittsburgh were planning their tunnels, plans were also being made for the Holland tunnel in New York, and the New York and New Jersey Commissions, in coöperation with the Bureau of Mines, went into a very extensive program of research work. I believe it was on the basis of that research that those in charge of the Pittsburgh tunnel decided to use two units.

I believe that the longitudinal system of ventilation of tunnels has its limitation. In this particular instance, the location was favorable for such a system, because the tunnel is through a mountain and as many shafts as necessary can be provided. If such a tunnel were under a stream, it would not be possible to do that, especially if the requirements of navigation were rigid. I am inclined to believe that the length of the section used in Pittsburgh is about the maximum that could be safely used in practice, because the longitudinal movement of air in a vehicular tunnel may be objectionable under certain conditions, especially in case of fire.

With longitudinal movement of air, the ventilation will be affected by the outside wind and at Pittsburgh, I know, they constructed baffles to overcome this difficulty. I am interested to learn how effective these baffles have been under actual operating conditions.

L. W. HUBER.—The wind baffles have been very effective; in fact, they have been entirely successful.

O. SINGSTAD.—You find that a head-on wind does not add to the load on the fan motor.

L. HUBER.—Not appreciably. The sides of the tunnel have been extended for 66 ft., and there is no roof, so that the air coming out of the tunnel meets the outside air and together they rise through the opening. So far as I know, that has been entirely satisfactory.

F. HAAS, Fairmont, W. Va.—What is the effect of a head-on wind of 40 miles an hour against one of the intakes of this tunnel? Mr. Huber mentioned a water gage of 1 in. A 40-mile gale, which is not unusual, will overcome that one inch and I can readily see how it would reverse the current. The explanation that the incoming air and outgoing air would meet and immediately go up together would not hold. Air does not act that way. I think the major current will overcome the other current, and in many cases the ventilation will be reversed. I can even go so far as to imagine a balanced condition in which it would be impossible to get rid of any of the bad air.

L. W. HUBER.—There is provision for variable speeds on the fan. As Mr. Haas mentioned, the head-on gale of 4000 ft. per min. probably would ruin ventilation in the tunnel almost entirely, but the speed of the fans can be increased two and one-half times, which would indicate a water gage of something like six inches.

F. HAAS.—What regulations have you in regard to changing the speed of the fan when these wind conditions occur?

L. W. HUBER.—At present, I do not know how such a situation would be met. At first, a carbon monoxide recorder was used in the tunnel and as soon as the carbon monoxide content got beyond the permissible limits, someone in the tunnel communicated with the power house or fan house and the fans were speeded up to give more ventilation, but I believe that those recorders are no longer used. It was found that the peaks came at regular times, and according to traffic conditions, therefore the speed is increased at regular hours.

F. HAAS.—Mr. Huber speaks of the exhaust running 70,000 cu. ft. per min. with 0.6-in. water gage. It appears to me that this power consumption is excessive.

LOUIS HUBER.—There is a 200-ft. shaft up to the exhaust fan and about a 60-ft. stack also, in addition to the tunnel.

D. HARRINGTON, Washington, D. C.—What is the area of the shaft?

L. W. HUBER.—One hundred square feet.

T. FRASER, Harrisburg, Pa.—A velocity of 800 ft. per min. for half a mile would generate a much higher water gage, I would think.

C. EVANS, Scranton, Pa.—I would suggest that this excessive water pressure is perhaps due in part to the right turn at the junction of the shaft and the tunnel itself. We mining engineers should learn something from that.

F. HAAS.—I can only answer that by a little experience. We are ventilating a mine now which has frequent turns. It covers a square mile or more and we are producing the first 100,000 cu. ft. of air with 0.16-in. water gage.

O. SINGSTAD.—I do not wish to criticise the tunnel but it seems to me that the lighting should have been arranged so as to illuminate the arch. I believe, also, that the appearance of the tunnel, beautiful as it is in cross-section, would be materially improved by tiling. That would have added not only to the pleasing effect, but also to the sanitary condition of the tunnel.

G. S. RICE, Washington, D. C. (written discussion).—Mr. Huber has given an interesting description of the ventilation of the Liberty tunnels. The system adopted is so different from that of the Holland tunnels, described last year by Ole Singstad, and embraces also such unique ventilation features, that study of the technical results is of great value to the student of underground ventilation methods. There are several points which awaken questions but before bringing these up I would like to add to Mr. Huber's statement regarding the amount of carbon monoxide given off by automobiles the comment that data on that point were obtained prior to the construction of the Liberty tunnels by laboratory physiological tests carried on by Dr. Yandell Henderson of Yale, and the quantity and kind of gases in the exhaust of gasoline motors was carefully determined by tests at the Bureau of Mines, made under the supervision of A. C. Fieldner.

Mr. Huber points out that there are certain natural ventilation effects due to the elevation of the portals of each tunnel, as well as to the wind pressures. Perhaps the difference of temperature on the two sides of the hill penetrated by the tunnels may make one of the principal natural impelling agencies. Mr. Huber says that the natural causes produce velocities of the air currents from zero up to 800 ft. per min. He adds, however, that the "induced ventilation tendency set up by the traffic is greater than the ventilation tendency set up by natural conditions" (when there is no artificial ventilation). Whether this is true or not at any particular time would seem to depend on the amount of traffic. For example, he states: "100 to 200 automobiles per hr. traveling at 30 to 40 miles per hr. will reverse an air travel of 200 to 300 ft. per min." Converting Mr. Huber's figures to the same time basis, there would be 3 to 6 automobiles in the tunnel at any one time, which, traveling at the rate of 2640 to 3520 ft. per min., would reverse a natural opposing air current of 200 to 300 ft. per minute.

In describing the power-plant air shaft and removal of vitiated air after it had traveled through half the length of the respective tunnels, Mr. Huber states that this method "reduces the period of contamination by one-half and lessens the frictional resistance considerably"—as compared with "through" ventilation, that is, without the upcast and downcast shafts in the middle. It had been my understanding that a portion of the air in the first half of each tunnel is carried on into the second half of the tunnel; if so, the contamination in the second half would tend to be somewhat higher than that of the first half. He quotes from a paper by Mr. Harvey, given before the Engineers Society of Western Pennsylvania, which states that "the air is there drawn out through the exhaust compartment of the vertical shaft."

In conferences held with Bureau of Mines officials and the consulting engineer, Mr. Neeld, and others while the tunnel was under construction, we were informed that it was expected from experiments with a model made in Milwaukee that from three-quarters to two-thirds of the air in the first half of the tunnel would be drawn out by the exhaust shaft and the other one-quarter to one-third would be carried on and be joined by sufficient fresh air from the downcast shaft to replace that which had been exhausted. The questions involved were studied by Mr. Paul and associates, and his discussion appears later in these pages.

The data given by Mr. Huber regarding the fans are most interesting. It appears that the fans were all of the same size and capacity, two blowing fans on each tunnel and two exhaust fans. Mr. Huber indicates that in practice the air pressures required in each of the blowing fans is 1.7 in. water gage and that required for each of the exhaust is only 0.72 in., although the load would appear to be equal. Subsequent to reading Mr. Huber's paper, in company with Mr. Singstad, I visited the tunnels and was impressed that a part of the greater load on the blowing fan was due to the restriction of the nozzles at the foot of the downcast shafts. The other part probably is due to the larger volume of air in the second half from the air passing the upcast shaft.

The use of forward-curving multivane fans in parallel gives an opportunity to study the question that was discussed at the meeting of the Ventilation Committee in 1926; i. e., that forward-curving fans when operated in parallel do not draw equal amounts of air from the upcast or exhaust duct. One tends to take more air and do more work than the other. This is suggested in the discussion of Mr. Harvey's paper before the Engineers Society of Western Pennsylvania. He says "when two fans of the type used on this work draw air from a common chamber, they will not divide the work equally, even when they are the same size and are running at the same speed. One or the other will, as commonly expressed, 'hog the road.' At one time this will occur to one of the fans and at another time to another fan; thus two fans operating in parallel might result in a fan motor being overloaded from time to time."

It is hoped that sometime further quantitative tests will be made to determine the reasons for this situation, and whether it would be true if the fans were kept at a uniform speed and automatic operating shutters were placed in the intake duct of each fan.

Another important question arises as to the degree of efficacy in the device at the Liberty tunnel of the so-called "windbreaks" at the exit portals of each of the parallel tunnels. This arrangement, if it is sufficiently successful, might be used with great advantage at all tunnel entrances and drift mouths in which mechanical ventilation is employed. Mr. Huber says that in practice the windbreaks seem to be very effective, but he does not give figures.

W. P. Yant, chemist of the Bureau of Mines, in a discussion which follows, gives data bearing on the effect of the windbreaks when the air in the tunnel is moved by the outside wind pressure and other natural causes, and by the movement of automobiles and trucks. These data were obtained before the mechanical ventilation equip-

ment was ready to use. Different relations might occur, when there is mechanical ventilation, so it is hoped that the tunnel authorities will permit tests to be made. Definite data would have distinct value to mining men as well as to tunnel engineers.

J. W. PAUL, Pittsburgh, Pa. (written discussion).—Reference is made in Mr. Huber's paper to the volume of air that may pass by the upcast shaft and pass through the tunnel. As no figures are given to indicate what this volume may be under operating conditions, the value of the paper may be increased if certain data are given concerning the induced volume of air passing the zone of the upcast and downcast shafts. This discussion is offered for the purpose of supplying some definite data on that phase of the ventilation of the Liberty tunnels.

Prior to the installation of the ventilating appliances at the Liberty tunnels, A. D. Neeld reported that tests conducted at Milwaukee with a model tunnel indicated that a relatively large percentage of the air entering the portal would pass by the upcast shaft and through the second half of the tunnel. After the Liberty tunnels had been equipped with ventilating fans, as described by Mr. [Huber, I had the privilege of conducting some measurements of the air in one of the Liberty tunnels, with the assistance of three of the engineers of the Bureau of Mines, and with the cooperation of Mr. Neeld. These measurements were conducted during the latter part of August, 1924, at a time when traffic was suspended in the east tunnel and when there was no prevailing wind. The velocity readings were taken with a calibrated anemometer which traversed the cross-section of the tunnel. A 9-ft. rod was used as its support and it was manipulated from a 9-ft. stepladder. Duplicate readings were made at each reading station, each over a period of 4 minutes. The locations of the reading stations were as follows: 312 ft. south of the upcast shaft; 38 ft. north of the upcast shaft, and 508 ft. north of the upcast shaft. The nozzle of the downcast shaft is 80 ft. north of the downcast shaft.

Readings were taken under the following conditions: (1) Exhaust and supply fans running at full speed; (2) exhaust fans running half speed and supply fans running full speed; (3) exhaust fans stopped, upcast shaft sealed off, and supply fans running full speed. Under condition (2) one fan was shut down and the other run at full speed to give the half-speed condition. Results are shown in Table 1.

TABLE 1.—*Measurements of Air in Liberty Tunnels, August, 1924*

Conditions	Rate of Fans R. P. M.		Air Velocity, Cubic Feet per Minute					Per Cent. of Air Entering Portal That Passed Up- cast Shaft
	Ex- haust	Pres- sure	Air Enter- ing Portal	Air Going Up Shaft	Air Passing between Shafts	Air Coming Down Shaft	Air Going to Exit Portal	
Condition 1 All fans running.	101	145	518,000	295,000	223,000	282,000	505,000	43
Condition 2 One exhaust fan Supply full speed	101	145	427,000	145,000	273,000	283,000	555,000	63 9
Condition 3 No exhaust fans, up- cast shaft sealed. Supply fans full speed.		145	311,000		311,000	289,000	600,000	100

At the time these readings were taken no pressure-gage readings were available.

Under Condition 1, with all fans running, it may be noted that the upcast air travels two compartments having a combined cross-section of approximately 286 sq. ft. and the air has a velocity of 1031 ft. per min. whereas the downcast air travels two compartments having a combined cross-section of approximately 156 sq. ft. and the air has a velocity of 1807 ft. per minute. As the downcast shaft compartments terminate in a nozzle with a restricted outlet, the pressure fans evidently will work under a greater load than the exhaust fans in moving the same volume of air.

W. P. YANT, Pittsburgh, Pa. (written discussion).—While I have no data from which to draw conclusions as to the effectiveness or ineffectiveness of the “wind-breaks” at exit ends of the Liberty tunnels when they are operating under mechanical ventilation, I will contribute certain information which has a bearing upon the windbreaks when mechanical ventilation is not in use. It was my privilege to secure this information through the assistance and coöperation of Mr. Neeld and the County of Allegheny before the installation of the ventilation system which Mr. Huber described. The tubes were completed approximately six months in advance of the ventilation system and, in response to the demand of the public, they were opened

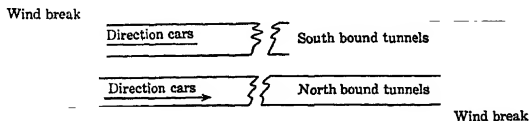


FIG. 8.—POSITION OF WINDBREAKS WITH REFERENCE TO WIND DIRECTION AND MOVEMENT OF CARS.

for use, the ventilation depending on that arising from natural air currents and that induced by the movement of the cars through the tunnel. Throughout this six-months' period a constant vigilance was kept on the safety of the tunnel through observations and air analyses and when an unhealthy or dangerous accumulation of exhaust gases was indicated the tunnel in which it occurred was closed to traffic. In connection with [this control, a wide variation of conditions was observed with respect to the ventilation created by natural air currents, which always had the same direction in both tunnels, and that induced by the car motion, which was opposite in the two tubes and which was additionally varied by not only an unequal number of cars in each of the respective tubes but by a varying number per unit time in a given tube. This produced a range of conditions that ran the entire gamut from zero to several hundred linear feet per minute velocity in the same direction in each tube from natural pressures; from zero to several hundred linear feet per minute in opposite directions in each tube due to number and speed of cars; and finally zero velocity produced by a balance between the natural ventilation and that induced by car motion when in opposite directions. This latter created the most favorable conditions for accumulation of gas but due to the cars traveling in opposite directions it never occurred in both tubes at the same time (Fig. 8).

During these observations daily readings were taken at a morning and evening period (usually 30 minutes) when both tubes were closed to traffic. The average results for a representative period are shown in Fig. 9. It will be noted that the velocities are quite similar in each tube. The variations may be due in part to natural changes which occurred during the time of taking measurements in each of the two tubes. However, in general there was a tendency for a lower velocity to occur in the tunnel that had baffles at the opening facing the wind. On the other hand, however, the baffles were not sufficiently effective under the conditions to prevent the reversal of the air current by wind facing the movement of the cars, even at times

when the number of cars was sufficient to induce a velocity of several hundred feet per minute in the absence of opposing wind. An outstanding example of this is shown in Figs. 10 and 11, which represent results obtained on Sunday, May 18, 1924. Fig. 10 represents the amount of traffic and air velocity in the southbound tube and Fig. 11 the same data for the northbound tube.

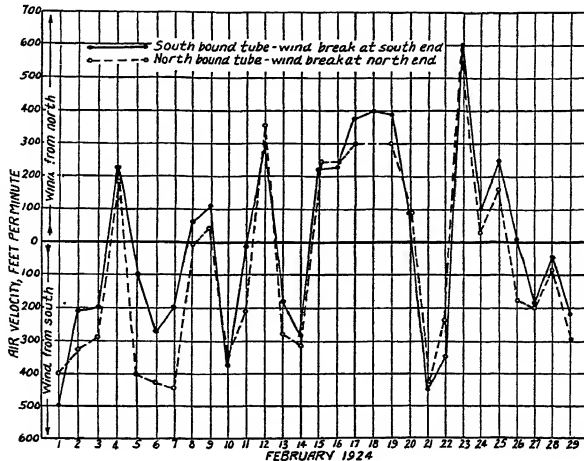


FIG. 9.—DAILY AVERAGE WIND VELOCITY THROUGH TUNNELS AT 11 P. M., WHEN TRAFFIC WAS SUSPENDED FOR 30 MINUTES.

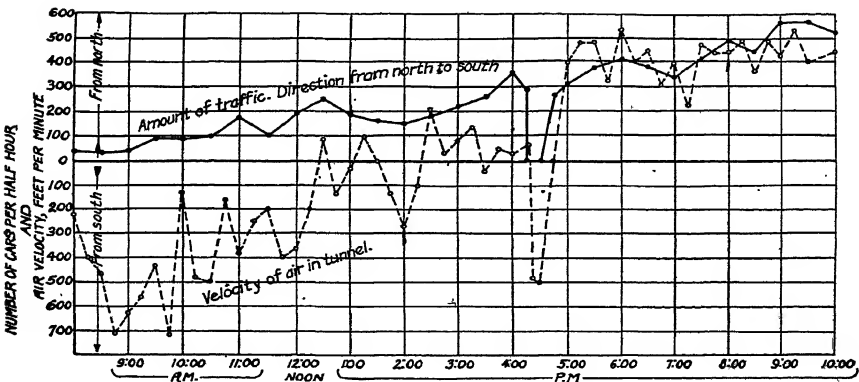


FIG. 10.—VELOCITY OF AIR IN SOUTHBOUND TUNNEL. WINDBREAK AT SOUTH EXIT.

It will be noted in Fig. 10 that with a low amount of traffic the air velocity was 200 to 700 lin. ft. per min. in a direction (north) opposite to the cars, an amount of traffic that under conditions of no natural air movement would induce 200 to 300 lin. ft. per min. in the direction of car movement (south). As the volume of traffic increased the tendency was to reduce the opposing natural wind velocity and at times to reverse it. The previously described balance between natural and car-motion

effects was obtained, with the result that a dangerous accumulation of gas occurred and it was necessary to close the tube to traffic at 4:15 p. m. With the stopping of the cars, the direction of air movement and velocity reverted to that prevailing in the morning and the tunnel quickly cleared of exhaust gases. When the tube was opened to traffic fifteen minutes later, the larger number of cars immediately reversed the wind and created adequate ventilation for the tunnel. This complete reversal of direction and maintenance of good ventilation can in part be attributed to the number

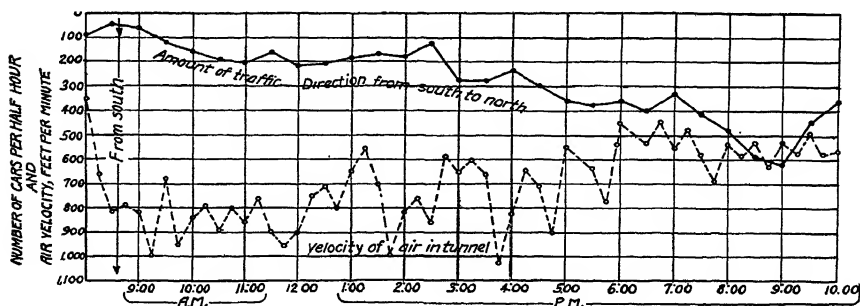


FIG. 11.—VELOCITY OF AIR IN NORTHBOUND TUNNEL. WINDBREAK AT NORTH EXIT.

of cars and in part to a decrease in the opposing force from natural cause during the period. The latter is manifested in the decrease in the velocity in the northbound tube (Fig. 11) even though the number of cars increased, otherwise the velocity should have been augmented, as both the car motion and natural velocity were in the same direction in this tunnel. Fig. 11 shows that the effect of the natural wind velocity moving in the same direction (north) as the cars is to increase markedly the velocity induced by the cars.

W. A. WELDIN, Pittsburgh, Pa. (written discussion).—During the designing and construction of the Liberty tunnels my firm⁶ held the position of consulting engineers to the County Controller and was in touch with the work of Mr. Neeld, the designer. In that connection a statement of some of the items that were considered at that time, particularly as to traffic, and a comparison with actual experience since may be of interest.

Mr. Huber states that with both traffic lines in one tube full of cars spaced 100 ft. apart, there is required 280,000 cu. ft. of air per min. in each tube, and that the number of cars assumed to be in the tubes at any one time is 113.

If the cars so assumed were moving at 15 miles per hr., the output of the tunnels in any one direction would be about 1600 vehicles per hr. A traffic count made by the city Bureau of Traffic Planning in April, 1927, shows that the largest concentration of traffic in the entire city was on Second Ave. between Smithfield and Wood Sts., when 9091 cars, westbound, were counted in 10½ hr. These were in two moving lines. The maximum hour shows 1058 or 123 per cent. of the average. The greatest artery leading into the congested district is Bigelow Boulevard with 8460 cars in one direction in 10½ hr. The Boulevard of the Allies had 6250. The 1600 vehicles per hr. assumed above, if the traffic has the same characteristics as to peak hours, would give the tunnel a capacity of over 13,000 vehicles a day, or more than double the present traffic on the Boulevard of the Allies.

⁶ Blum, Weldin & Co.

Since the territory tributary to this tunnel is not of such nature that it can readily be prepared to support in any very short time a population comparable to that which feeds either of the boulevards mentioned, it is evident that there is here a very wide margin of safety. The 1927 count, 3 years after the completion of the tunnel, showed 5200 vehicles inbound and 4200 vehicles outbound in $10\frac{1}{2}$ hr. In addition to this large margin, we are informed that by using both motors on each fan, it is possible to secure in an emergency 350,000 cu. ft. of air per min., which gives another 25 per cent. margin over the 280,000 assumed.

The above has to do, of course, with probable traffic growth in the near future. As to the ultimate capacity, according to Olmsted, Bartholemew and Chaney, Los Angeles city planning traffic experts, a traffic capacity curve indicates that approximately 2400 vehicles per hr. pass a given point at the rate of 15 miles per hr., making 4800 for two traffic lines. At 15 miles per hr., the transit of the tubes requires about 4 min., only half of which is spent in one ventilating current. Most vehicles pass through in much less time. Since the safe dilution of 6 parts in 10,000 is based upon exposure for 1 hr., there is here another large factor of safety, especially for motorists.

It is probably wise to provide for the largest air flow to be anticipated in the remote future, so far as to proportion the shafts, nozzles and other permanent structures to them. It seemed to us at the time, entirely unnecessary to provide fans and motors for such maximum air volumes, seeing that the need for them, when it does arise, will be many years in the future. Economy would seem to dictate the installation of additional or larger units when needed, rather than equipping the entire plant at first to provide for traffic which cannot mature for many years.

It was regarded as possible, however, that a blockade in the tunnel might produce peak conditions at almost any time. Assuming that the tunnel is entirely full of automobiles with only 5 ft. clear between them, the vehicles being distributed equally among the six classes tested by the Bureau of Mines, all with engines idling and giving off 35 to 66 cu. ft. of CO per hr. as determined by the Bureau's experiments, the dilution requirement would indicate the need for 277,000 cu. ft. of air per min. This would indicate that the 280,000 cu. ft. per min. assumed would be ample for any condition that can possibly arise.

The interesting problem of proper regulation of the ventilation was very much discussed. Mr. Neeld considered it necessary to have four rotary converters, direct current motors and controllers designed for as many as 20 different speeds. It seemed to us from a study of available curves showing the hourly variation of traffic, that such close regulation would be entirely impracticable. Moreover the danger of delay, blockade and fire in the tunnel would make sudden increases necessary. It is highly doubtful if even with many more police than are now in use, it would be possible to discover such a change in conditions, telephone it to the fan house, increase the speed of the fans, and have the fresh air arrive at the critical point in time to prevent persons from being overcome. This is especially true of fires or blockades occurring at one of the times of minimum ventilation.

It is also proposed to vary the output of the fans according to the vagaries of the capricious winds. All this regulation is to be accomplished through the intervention, as we understand it, of one motorcycle policeman for the entire 2 miles of tunnel. We are not convinced that the arrangements for regulating the ventilation even for the flow of ordinary traffic are adequate. Newspaper accounts of at least one pedestrian being overcome seem to support this fear.

Even though it can be shown that the curve of daily flow of traffic has a characteristic form, it is not necessarily true that it can be followed at all closely by variations in air flow. At any minute, a procession, a crowd from an athletic or other event, a fire, a blockade, or a wind may utterly upset the schedule. For this reason, it seems to us that the regulation should be aimed to be provide for most of the fluctuations by

circulating at all times very much more air than required by volume of traffic alone so that emergency speeding up would be confined to extraordinary occurrences. Also that upon the unusual happening, the excess of air regularly provided would probably care for the earlier stages of the emergency, thus giving time to apprehend the presence of dangerous conditions, to ask for more air, and to get it. Such a view of regulation would indicate the need for only a very few changes of volume which could be produced by far simpler apparatus than that adopted. In fact, we are informed that only two air quantities are used at present.

In view of the almost complete absence of policing, we regard as alarming the statement that the use of the CO recorder has been discontinued. It seems to us that this device, properly installed, might enable the fan operators to observe from time to time the actual conditions and give them a real chance to meet promptly the need for more air when it occurs.

Report of Committee on Metal Mine Ventilation

PRESENTED BY D. HARRINGTON, CHAIRMAN

(New York Meeting, February, 1927)

THE ventilation of metal mines is becoming more of a fundamental requirement than it was 10 or even 5 years ago. There are many reasons for this, and in general they are based on economy or efficiency rather than health or safety, yet the latter enters into the picture in at least some instances. One Montana metal mine which had installed mine fans and a general ventilating system shut down the fan when there was a period of cessation of work; within a few weeks it was found that the underground timbers were becoming so covered with fungus that the fans were put into operation and thereafter continued in operation whether the mine was worked or not. The fungus soon disappeared and the cost of operating the fans was held to be far less than would be the numerous losses from timber decay which⁷ quickly started when cessation of air flow was encountered.

A large iron ore mine in Michigan had much difficulty in holding men in a slicing region due to foul air under certain surface conditions as to direction of wind. Upon the placing of a moderate sized fan on the surface with some doors, stoppings and regulators underground, not only was the bad air condition removed but there was found to be a decided increase of tonnage per man per shift and timber decay was retarded to a very considerable extent.

This particular mining company was rather inclined to be angry when I called attention to some of the what I considered very bad working conditions underground, and the mine superintendent for about a year and a half was on anything but friendly terms with me. Later on when the effect of the placing of the fan was found (the fan had been placed over his protest), that man came through very nicely with a letter and congratulated me on the recommendation made and congratulated himself and the company on the conditions obtained after the installation of the fan.

POISONOUS FUMES AFTER BLASTING

A copper mine in the Southwest with an ore of a high sulfide content has much difficulty due to poisonous fumes after blasting, and the fan installation is as elaborate as those of many of the most nearly up-to-date coal mines, and coal mine splitting and regulating practices are being

used to a remarkable extent. The regulators are of concrete construction, with the door portions of steel and are the most efficient ventilating regulators I have ever seen in a mine, coal or metal. The mine has such dangerous gases at blasting time that there is an exact time of blasting for each level, and to make certain that all parts of the mine have the same time in accordance with their watches, Western Union clocks are installed on numerous levels of the mine.

In addition to the poisonous gases to be handled by the air currents in this mine there are at times explosions or ignitions of dusts or gases or both, and on each level there are rescue chambers provided with tight doors and with water, compressed air and gas masks in each one of these rescue chambers. This mine less than a year ago had what was certainly an ignition of sulfide ore dust that carried flame or at least burning dust several hundred feet (at least 700 ft.), and in its path killed three men and very seriously burned a fourth. The ignition was caused by the blasting of a raise in sulfide ore, the sulfur content of the ore being around 40 to 45 per cent., I think about 43 per cent.

METHANE

Several non-coal mines have at times encountered methane, the explosive gas of the coal mine, these mines being producers of iron, copper and other ores. One of these mines now uses the coal miner's electric cap lamp in parts of the property and all of them are forced to use mechanical ventilation.

In driving tunnels for city water purposes, there have been numerous instances of methane occurrence, and where the strata are by no means those of a coal region. In one tunnel there were nine men killed in 1926 due to an explosion of methane, ignited probably by electricity. In such instances coal mine ventilation methods as well as the usual inspection and precautionary practices and installations for gassy mines are required. In this particular tunnel instance, the operating company went from blasting with 40 per cent. gelatin to the use of permissible explosives. They tore out practically all of the electrical wiring, installed enclosed switches where they could get them, doubled their ventilation of the tunnel, tried to concrete sections of the tunnel which were giving off gas, and employed a Bureau of Mines foreman miner as safety man to inspect the tunnel daily and to take daily gas samples of the various parts of the property.

INERT GASES

Some metal mines have more or less frequent outflows of inert gases such as nitrogen and CO₂, and numerous lives have been lost from these gases, due to these gas occurrences. One of the districts which has the

nitrogen gas occurrence is that of the Tonopah region, described by Mr. Pickard.¹

In Cripple Creek, Colo., some of the mines use surface fans to force air into the mines, allowing the air no outlet except that which it finds through the strata, simultaneously forcing back into the strata the inert gases which would otherwise fill the mine workings and cause death. As a matter of fact, CO₂ gases have filled the mines in this district and gases have flowed out of the drifts just like water flows out.

At the present time the Bureau of Mines is making a study of the gas conditions of the Cripple Creek region, largely with the intent to try to ascertain the method of formation or the method of occurrence of this gas. They have done a considerable amount of experimental work and have taken some of the rock strata from these mines in the Cripple Creek district and placed them in sealed containers with some of the water from that district. They find that in a comparatively short length of time practically all of the oxygen left in the container is absorbed or probably is chemically combined with the constituents of the rock and the water, and there is practically nothing left but CO₂ and nitrogen and that is done in a remarkably short length of time.

MINER'S CONSUMPTION

There has been much progress in the past 10 years in metal mining ventilation, particularly in the West in the states of Montana, Idaho, Nevada and Arizona, and to a less extent in Colorado and California. Much of the ventilating is due to the necessity of removing dusts, fumes or gases, but a large amount of the attention has been paid to the matter of securing greater efficiency in hot workings by methods of air ventilation. The main fans are usually found on the surface, electrically driven, and in general so placed that direction of air currents may be reversed if desired. For underground distribution to faces, some mines use coal mine splitting systems, but in general ventilation is best accomplished in metal mines by small blowers in tubing either of galvanized iron, canvas or other flexible material. It has long been known that one of the best preventives of miner's consumption is the use of adequate air currents. Up to the present time miner's consumption has not been looked upon very seriously in the United States from an economic point of view, although it is recognized as the source of much sickness and of many deaths, especially in certain districts.

There has during the past year, however, arisen a situation in metal mines of Ontario, Canada, that should cause the metal mining men of the United States to take notice. Miner's consumption, or silicosis, has been designated as compensable in Ontario in essentially the same manner

¹ B. O. Pickard: *Underground Air Conditions and Ventilation Methods at Tonopah, Nev.* See p. 107.

as our accidents are, and to date there have been placed against the Ontario mines somewhat less than a quarter of a million dollars of assessments or of obligations to pay workers due to silicosis awards. As a result of this condition most of the mines of Ontario have shown a decided activity toward the installation of ventilating systems as well as of water blasts, sprays and other methods and devices looking toward the removal of dust and gases from the places where the men work or of preventing formation of those dusts or gases where feasible.

Many metal mining districts in the United States should take warning of the Ontario situation and remove from the mines the conditions suspected or known to cause miner's consumption. Foremost of the precautionary measures is the operation of an adequate ventilating system.

MINING VENTILATION LITERATURE

During the year 1926 there were several notable contributions to the literature of ventilation of mines with data largely if not wholly applicable to metal mines. Probably the most valuable of these articles or books which have come out during the past year is the one by Professor Weeks of the University of California. It is called "Ventilation of Mines,"² and in my opinion the title should have been, "Ventilation of Metal Mines," or possibly "The Physics and Mechanics of Metal Mine Ventilation." Professor Weeks has gone into the various phases of the physics and physiology of mining ventilation, and particularly of metal mine ventilation to an extent that I do not think any other author has done previously. He has called attention to nearly all of the up-to-date equipment and up-to-date methods of ventilation in mines, metal mines particularly, and has given references, a bibliography, to the metal mining people in a way that I do not think they can get in any other place. He has called the attention of the metal mining people to a number of things which very few of them realize affect them and which I think they can very well take up to advantage to themselves.

DISCUSSION

J. J. RUTLEDGE, Baltimore, Md.—Mr. Harrington referred to a matter that perhaps Mr. Tillson has met with in his practice. Some years ago when I was at work in the metal mines in New England mining big sulfide orebodies at great depth, I used the first water Leyner drill that was ever used east of the Rockies, I believe. At that time we were using about 40 per cent. dynamite and doing a good deal of drifting, stopping by the Cornish underhand system. If the word went around the mine that so-and-so was going to fire the drift, nobody remained in the mine. Nobody who knew in advance of the shot being fired remained in the mine workings. The fumes resulting from the blasting of the dry holes had such a disastrous effect on the lungs of the miners that there were ironclad orders that compressors were never to be

² W. S. Weeks: Ventilation of Mines.

stopped while any men were in the mine. Of course, the water Leyner did away with this trouble.

I estimate roughly that that company alone lost about 50 per cent. of the efficiency of their men from that cause.

B. F. TILLSON, Franklin, N. J.—I understand that the legislature of the state of New Jersey (possibly it is passed by now) has up for consideration a bill including silicosis among diseases compensable by industry, and I do not see why it should not be. It seems to me that industry or the consumers should pay the cost of the production, whatever is necessary, and not the individual, although the elaboration of this principle may be developed to absurd and unfair claims. The question will arise that suppose a man gets his feet wet and dies of pneumonia while working in an industry, should he be compensated? However, such a disease may originate from causes outside of the industry, and so it is hard to prove it an occupational hazard, but with silicosis the source of ill health may be definitely determined.

There is a question that Mr. Harrington raised that is rather interesting to me. I wonder if he can give me a little more information on it. He spoke of the increase of fungus growth being dependent upon the air supply, *i. e.*, the air current, and I presume that it is due to increased propagation of the tissues of the fungus material with the high percentage of carbon dioxide. If so, in the metal mine where, particularly, is that carbon dioxide coming from if there is nobody working in the mine, and the mine is idle, as I understood the example that you referred to.

D. HARRINGTON.—We nearly always have timber in the mines, and timber is decaying, and of course timber is composed largely of carbon. You get an enormous amount of carbon dioxide. We are not very familiar with the exact conditions under which fungus does and does not grow. There have been studies made of it and there are studies being made of it at the present time. We know that fungus seems to thrive where the temperature is fairly high, yet not above 100° F., and where the air is highly humid and where there is not too much air flow or air circulation.

On the other hand, there are a number of authorities who believe that the fungi, which are largely bacteria, will not live in an atmosphere practically free of oxygen, but there is no question on earth that sluggish ventilation—lack of air circulation—in mines is very likely to result in the growth of fungus and the growth of fungus is likely to result in decayed timber. One of the most striking examples of that is one of the iron ore mines in Birmingham, Ala. Five or six years ago this mine was shut down. Before it was shut down one slope having a slight pitch had been driven for a distance of about 2000 ft. without any connections. A parallel slope driven about the same distance had a number of gangways, or practically rooms. Where the gangways or rooms were driven there was a certain amount of natural circulation. In the long level that had no connections, there was no circulation whatever, and at the end of about 18 months when the mine was reopened it was found that practically all of the timber in the 2000-ft. slope without connections had decayed and fallen of its own weight. Just before the shutdown this entire 2000 ft. had been timbered, and heavily timbered at that. On the other hand, when they went into the slope that had the gangways or rooms off it, the timber was practically intact, there was air circulation in that and the timber had not been decayed at all. There was no difficulty whatever in convincing the manager of that mine that the mine should be ventilated and the mine was ventilated.

B. F. TILLSON.—I raised the question because I have not noticed a striking difference in the fungus growth on oak and chestnut timbers in our mines, depending upon the location. Even in the dead-end headings where we do have fungus we have it just as much where there seems to be a natural draft and circulation in the drift.

I wonder whether it is not possibly due to different species of timbers or different species of fungi that happen to be native to that habitat, because I have not noticed any material difference.

D. HARRINGTON.—I think probably there is something in that too, and before leaving the subject, I would like to give another example. About a year and a half ago one of the coal mines of Utah was closed, and in closing it they shut off the fan as well; they practically sealed the mine. In about 2 months I was asked to go through and make an inspection. The fan was started up one night and the next morning the safety man and I went through the mine. There was practically not one timber in that mine that had not been thoroughly whitened with fungus in the period that the fan had been shut down—a period of $2\frac{1}{2}$ or 3 months. We immediately decided that it would not pay to stop the fan, and while the electrical bill was heavy, the fan was operated every night at a time when the peak load would be at its lowest. At the end of about 2 weeks' operation of the fan we made another inspection of the mine and found practically all the fungus dead; it had flattened out and had almost disappeared, and from that time on there was no difficulty with the timber.

I have been told that this mine has been reopened and the timber was found intact. The management were thoroughly convinced of the necessity for keeping the fan going whether or not the mine was in operation.

Degree of Liberation of Minerals in the Alabama Low-grade Red Iron Ores after Grinding*

BY WILL H. COGHILL,† ROLLA, Mo.

(New York Meeting, February, 1927)

IN this investigation, the low-grade red iron ores of Alabama were examined by the use of "heavy solutions."¹

Ores are generally ground preparatory to concentration and as long as the concentrators make suitable products the effectiveness of the grinding is likely to be unquestioned; but when suitable products are difficult to make, an investigation becomes imperative. In such a case a dissection of the ore might be made to find the cause of the trouble. Such a condition of unsuitable products exists in the low-grade red iron ores of Alabama. They are difficult to concentrate. To be sure it might be said that the concentration of iron ores is a new thing and that the ore-dressing methods are undeveloped, but a glance at the industry reveals an abundance of methods and devices for almost all imaginable conditions.

Concentrating machinery has not been tested in this investigation. When the properties of the ores have been gleaned, the machinery may be used more effectively.

ALABAMA LOW-GRADE RED IRON ORES

The Bureau of Mines has undertaken by laboratory methods to select the best means of concentrating the low-grade iron ores of Alabama. These ores constitute the parts of the ore bed left by selective mining. If more of the ore bed could be included in the current output, the mining cost would be reduced and longer life of the district would be insured. This advantage would be reflected by added permanence of churches, homes, schools and every undertaking in the commonwealth.

The red ores chosen for this study of effectiveness of grinding are too low in lime to meet the requirements of the blast furnace. If lime is

* See also J. T. Singewald, Jr.: Beneficiation of Alabama Siliceous Red Hematite. A. I. M. E. Pamphlet No. 1653-B (February, 1917).

† Metallurgist, Mississippi Valley Experiment Station, Bureau of Mines.

¹ A recent investigation of heavy liquids is reported by J. D. Sullivan in *Tech. Paper* No. 381, U. S. Bureau of Mines, on The Preparation and Properties of Heavy Liquids for Mineralogical Analysis.

added the burden would be too great. If silica could be removed an acceptable smelting product would result. The iron content is not far from that of the present smelting ore, but the silica content is too high. The appellation, ferruginous sandstone, is significant; the ores are oölitic and fossiliferous.

Five representative samples were taken. The average content is 33 per cent. iron, 13 per cent. lime and 23 per cent. insoluble. The average grade² now being smelted is 36 per cent. iron, 17 per cent. lime and 14 per cent. silica and alumina.

SCREEN ANALYSES

The screen analyses of the five respective ore samples are shown in Table 1. The same method of grinding was applied to each and they were made to pass 14 mesh. The Tyler scale was used and the screening extended to 325 mesh. Wet screening reduced attrition to a minimum.

The "weight per cent." when plotted would give curves similar to what is always expected when consecutive screens have openings bearing a fixed ratio. The largest weight per cent. in this case is very near the coarsest size because of stage crushing to avoid sliming. The five curves superimposed would nearly coincide, and the assays of iron, lime and insoluble show how nearly alike, chemically, are the respective ores.

In making a diagnosis the nature of the considerable amount through 325 mesh is the first indication of an important characteristic. Doubtless northern iron ores, which are being concentrated by the early log-washing methods would also show a large percentage through 325 mesh, but a chemical analysis would show it to be high in removable gangue. This is not so in the Alabama ores. The "iron, assay per cent." column shows that the finest mesh is richer in iron than any other size. For example, the composite sample of ore No. 1 has a tenor of only 36.5 per cent. iron, while the part passing the last sieve assayed 45.1 per cent. Hematite is the softest constituent and an attempt to practice log washing would put the richest part of the ore into the overflow. If any part could be thrown away it would be the coarsest. The iron grades up in the finer sizes, lime does not vary materially, and "insoluble" grades down; the finest is the richest in iron and the leanest in insoluble. The five samples are very much alike in these respects.

Table 2 shows the result of dissecting the samples size by size, into specific gravity increments. The iron, lime and insoluble contents are given. Phosphorus and sulfur have not been taken into consideration. The dissection was with heavy liquids.

² W. R. Crane: Red Iron Ore Mining Methods in the Birmingham District. *Trans.* (1925) 72, 157.

TABLE 1.—Screen Analysis of Five Alabama Iron Ores

Mesh	Weight, Per Cent.				Iron Assay, Per Cent.				Percentage of Iron				Lime Assay, Per Cent.				Insoluble Assay, Per Cent.							
	Ore No.	Ore No.	Ore No.	Ore No.	Ore No.	Ore No.	Ore No.	Ore No.	Ore No.	Ore No.	Ore No.	Ore No.	Ore No.	Ore No.	Ore No.	Ore No.	Ore No.	Ore No.	Ore No.					
	1	2	3	4	5	1	2	3	4	5	1	2	3	4	5	1	2	3	4	5				
- 14 + 20	13.59	18.76	18.70	18.28	18.25	32.9	31.0	20.0	30.9	21.5	12.2	16.1	15.1	17.7	14.6	11.1	12.3	18.0	14.0	25.2	33.7	33.9	14.8	35.1
- 20 + 28	13.45	20.17	19.54	16.96	17.15	32.4	33.1	30.1	32.4	26.0	11.9	18.2	17.9	17.1	16.6	11.2	11.2	17.4	12.9	25.9	31.1	20.9	13.0	31.5
- 28 + 35	14.08	16.48	18.01	16.28	16.00	34.6	35.0	32.3	33.3	23.0	13.4	15.7	17.7	16.8	18.0	11.0	11.4	17.5	11.4	22.8	28.0	27.7	14.7	28.6
- 35 + 48	15.20	11.48	13.09	12.12	11.55	34.9	36.4	33.3	31.8	30.2	14.5	11.4	13.3	12.0	13.0	12.9	13.4	17.9	11.9	20.3	25.0	22.4	14.3	26.4
- 48 + 65	9.80	5.96	6.29	6.60	7.10	36.7	38.1	34.0	30.9	27.2	9.9	6.2	6.5	6.4	7.2	12.0	13.5	17.7	13.8	17.1	12.1	20.8	13.7	26.5
- 65 + 100	8.28	6.99	6.29	6.40	7.50	35.8	40.1	36.0	31.4	26.7	8.1	7.6	7.0	6.3	7.5	12.7	12.3	17.1	13.0	16.3	18.0	16.9	14.2	29.3
- 100 + 150	5.34	4.56	3.49	3.96	4.05	38.6	42.8	38.5	32.4	26.5	5.7	5.2	4.1	4.0	4.0	11.4	12.0	17.8	15.6	17.5	15.9	14.0	29.4	
- 150 + 200	3.77	2.21	2.32	2.93	2.85	40.0	43.7	37.8	30.9	26.4	4.1	2.6	2.7	2.8	2.8	11.2	13.6	19.1	15.4	16.6	15.4	14.1	129.0	
- 200 + 270	2.72	2.28	2.32	2.84	2.50	43.2	43.0	41.0	32.8	27.4	3.2	2.7	2.9	2.9	2.5	9.0	12.2	18.4	13.8	17.0	14.1	14.3	28.2	
- 270 + 325	1.71	1.18	1.08	1.56	1.20	43.8	45.1	40.9	31.5	26.4	2.1	1.4	1.4	1.5	1.2	9.1	11.1	19.9	13.9	16.3	13.8	14.6	20.1	
- 325	12.06	9.93	8.87	12.07	11.85	45.1	48.0	42.2	33.3	28.5	14.9	12.9	11.4	12.5	12.0	8.0	11.9	17.4	17.4	12.8	12.1	11.6	12.8	22.3
Composite	100.0	100.0	100.0	100.0	100.0	36.5	36.7	32.8	32.1	26.8	100.0	100.0	100.0	100.0	100.0	11.2	12.1	17.7		19.8	25.7	24.9	14.1	29.2

TABLE 2.—*Specific Gravity Analysis of Five Alabama Red Iron Ores*

Specific Gravity Increments	Weight, Per Cent.			Iron Assay, Per Cent.			Percentage of Iron			Lime Assay, Per Cent.			Insoluble Assay, Per Cent.		
	Ore No. 1	Ore No. 2	Ore No. 3	Ore No. 4	Ore No. 5	Ore No. 1	Ore No. 2	Ore No. 3	Ore No. 4	Ore No. 5	Ore No. 1	Ore No. 2	Ore No. 3	Ore No. 4	Ore No. 5
-14 + 20 mesh															
Float on 2.90 sp. g.	19.0	20.6	27.0	17.4	34.4	5.7	7.7	6.3	4.3	5.7					
2.90 to 3.34	20.5	20.7	29.4	20.0	22.4	29.0	22.4	12.6	14.7	22.6	13.0	21.9	14.1	30.5	20.5
3.34 to 3.70	13.1	14.4	14.7	31.0	18.6	33.3	33.8	13.3	15.7	13.6	17.7	4.3	8.6	12.0	17.2
3.70 to 4.20	43.0	39.1	23.0	31.0	18.0	48.1	45.7	62.8	56.6	41.1	44.5	38.1	5.9	4.2	3.9
Sink in 4.20 sp. g.	4.4	5.2	5.2	6.7	0.8	59.4	57.0	6.2	9.3	11.2	12.2	2.1	1.9	2.5	4.1
-20 + 28 mesh															
Float on 2.90 sp. g.	100	100	100	100	32.9	31.6	26.6	30.9	21.5	100	100	100	11.1	12.3	18.0
2.90 to 3.34	20.9	18.4	24.0	17.4	34.6	5.3	5.9	6.1	4.9	6.3				14.6	33.1
3.34 to 3.70	19.9	21.1	24.3	17.4	25.3	19.7	21.2	20.0	7.9	3.9	10.3	19.9	20.1	17.3	28.1
3.70 to 4.20	11.6	13.6	13.4	23.2	10.5	33.3	34.0	34.0	33.6	34.0	12.1	13.5	16.7	10.7	28.0
Sink in 4.20 sp. g.	7.2	5.1	12.5	9.9	12.2	60.7	59.3	57.6	57.9	58.0	50.2	40.2	31.3	4.2	5.7
-23 + 35 mesh															
Float on 2.90 sp. g.	100	100	100	100	32.4	33.1	30.1	32.4	26.0	100	100	100	11.2	11.2	17.4
2.90 to 3.34	19.4	13.1	21.7	16.4	22.5	20.0	20.4	20.0	19.0	19.5				16.7	35.2
3.34 to 3.70	10.2	9.5	12.4	19.6	9.0	33.3	33.1	35.0	33.6	34.0	9.8	9.0	13.5	18.5	28.0
3.70 to 4.20	32.5	37.7	23.6	31.2	26.3	47.5	46.4	45.8	46.3	49.6	44.5	43.6	54.3	8.2	4.3
Sink in 4.20 sp. g.	13.1	15.2	19.9	14.1	13.7	60.0	58.7	57.7	58.3	59.5	51.5	52.5	52.4	1.4	1.6
-35 + 48 mesh															
Float on 2.90 sp. g.	100	100	100	100	34.6	35.0	32.3	33.2	30.2	100	100	100	11.0	11.4	17.5
2.90 to 3.34	19.9	8.5	21.3	23.2	27.7	4.5	4.1	4.8	3.3	4.9	2.5	0.9	3.1	21.6	34.1
3.34 to 3.70	19.8	26.8	18.5	16.6	22.3	20.2	21.5	9.7	11.8	19.5	11.5	11.7	9.5	26.5	28.5
3.70 to 4.20	7.8	13.0	11.9	17.0	9.3	33.0	35.0	33.6	33.3	33.2	7.7	4.2	12.0	18.0	10.1
Sink in 4.20 sp. g.	23.3	32.7	0.2	22.7	15.5	16.3	59.0	57.8	58.5	59.5	40.2	42.8	39.8	6.6	6.8
-48 + 65 mesh															
Float on 2.90 sp. g.	18.3	11.6	20.2	26.8	83.6	4.5	3.5	3.7	3.3	5.7	2.3	1.1	2.2	28.1	33.0
2.90 to 3.34	16.3	23.5	19.2	17.7	22.6	19.1	22.8	19.0	18.5	18.5	8.5	10.4	10.7	28.1	28.5
3.34 to 3.70	9.1	12.7	13.1	14.7	9.7	33.3	33.2	33.3	33.3	33.2	8.3	10.7	12.9	16.1	13.4
3.70 to 4.20	24.0	14.9	25.2	20.5	18.1	46.6	44.9	44.9	44.9	44.9	31.3	31.3	31.3	6.3	6.3
Sink in 4.20 sp. g.	31.8	37.3	32.3	30.3	31.6	58.7	57.7	59.8	59.2	58.3	50.8	56.6	39.8	8.6	8.9

- 65 + 100 mesh		20.1	10.9	22.5	30.2	34.5	3.8	2.9	3.7	3.4	4.4	2.1	0.8	2.3	3.2	5.7	29.7	31.4	37.0	24.9	28.6	13.1	15.6	24.5	16.2	7.7	35.2	20.4	39.5
Front on 2.90 sp. g....		14.4	19.5	16.5	5.26	4.20	4.4	20.5	19.7	17.9	19.9	8.2	10.1	8.9	8.8	19.7	20.6	24.9	28.6	13.1	15.6	24.5	16.2	7.7	35.2	20.4	39.5	14.8	39.5
2.90 to 3.34.....		22.0	69.6	4.3	11.9	8.3	32.7	51.4	33.4	34.1	134.8	20.1	38.2	3.9	12.9	10.8	15.0	15.0	5.0	7.4	13.2	10.1	13.4	14.8	25.6	8.0	8.9	8.0	8.9
3.34 to 3.70.....		20.9	19.2	14.4	8.5	44.6	5	43.2	24.7	0.45	7	14.1	22.7	21.5	14.6	8.1	0.3	2.7	5.9	4.0	14.1	14.0	12.0	14.4	8.0	8.0	8.0	8.0	8.9
Sink in 4.20 sp. g....		32.6	37.5	23.0	22.3	60.9		60.7	60.0	58.8	55.5	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100
- 100 + 150 mesh		100	100	100	100	35.8	40.1	36.6	31.4	26.7	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100
Front on 2.90 sp. g....		21.9	11.1	23.2	29.4	0.41	6	3.7	2.7	3.6	3.2	4.5	2.1	0.7	2.2	3.3	7.0	31.1	33.1	40.5	23.8	82.2	58.8	81.8	19.0	39.6	14.2	29.3	14.2
2.90 to 3.34.....		11.4	17.5	13.1	16.3	22.8	20.3	21.0	19.3	13.3	32.1	6.0	8.7	6.5	7.0	18.0	18.9	24.4	25.8	12.9	15.6	24.7	15.8	17.0	37.7	16.1	24.3	16.1	24.3
3.34 to 3.70.....		13.2	71.3	4.3	31.4	53.8	37.9	34.8	35.1	10.7	90.6	11.2	13.1	12.5	8.7	1.5	0.3	2.4	5.4	15.6	16.0	13.4	16.5*	7.6	7.7	9.5	7.6	7.7	9.5
Sink in 4.20 sp. g....		44.1	43.8	65.6	26.3	61.2	61.5	61.2	60.4	70.0	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100
- 150 + 200 mesh		100	100	100	100	38.6	42.3	38.5	32.4	26.5	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100
Front on 2.90 sp. g....		22.2	12.6	35.1	39.5	43.9	3.4	4.0	7.5	3.6	4.6	1.9	1.2	7.0	4.6	7.7	30.6	32.4	38.9	24.8	81.2	58.3	25.6	18.2	38.5	17.2	38.2	17.2	38.2
2.90 to 3.34.....		11.7	12.3	3.0	12.7	16.5	7.24	11.9	10.7	17.9	20.8	6.4	6.7	2.0	7.3	13.0	22.2	18.9	23.7	11.7	18.3	22.6	16.6	17.2	38.2	17.2	38.2	17.2	38.2
3.34 to 3.70.....		9.5	75.2	8.5	6.6	9.2	50.9	53.5	39.4	36.1	31.3	7.4	8.2	10.7	7.7	10.9	16.0	16.2	3.9	15.2	9.5	15.8	17.4	29.2	14.6	10.3	14.6	10.3	14.6
Sink in 4.20 sp. g....		49.6	44.5	65.5	37.9	61.5	62.5	62.0	60.3	76.0	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100
- 200 + 270 mesh		100	100	100	100	40.0	43.7	37.8	30.8	26.4	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100
Front on 2.90 sp. g....		22.4	0.0	31.0	41.4	49.5	3.9	6.4	4.0	5.2	2.0	4.8	5.0	4.8	5.0	9.4	29.7	34.1	38.9	24.8	51.0	58.0	16.2	25.2	19.2	37.7	25.2	19.2	37.7
2.90 to 3.34.....		7.3	6.2	8.1	8.3	5.9	18.8	3.6	19.1	18.0	28.0	3.2	0.5	1.5	4.5	5.6	20.7	20.0	23.8	0.5	20.5	58.0	16.2	25.2	19.2	37.7	25.2	19.2	37.7
3.34 to 3.70.....		4.4	63.8	5.6	5.1	10.1	33.6	45.6	6.80	9.38	5.28	7.3	4.9	4.2	6.0	9.9	13.5	15.0	15.0	1.0	0.3	15.8	13.7	17.8	30.4	17.8	30.4	17.8	30.4
Sink in 4.20 sp. g....		60.1	54.3	41.3	30.3	61.4	62.3	62.2	60.7	85.3	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100
- 270 + 325 mesh		100	100	100	100	43.2	43.0	41.1	32.8	27.4	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100
Front on 2.90 sp. g....		22.3	0.0	28.4	44.2	54.2	4.6	4.2	4.3	5.8	2.3	2.9	6.0	11.9	27.9	9	31.4	38.9	25.9	0.6	19.3	63.6	16.0	25.8	19.6	40.2	25.8	19.6	40.2
2.90 to 3.34.....		5.8	7.4	4.4	9.2	0.6	18.3	4.4	18.7	17.1	29.6	2.4	0.7	2.0	5.0	7.2	21.9	20.0	25.9	0.6	19.3	63.6	16.0	25.8	19.6	40.2	25.8	19.6	40.2
3.34 to 3.70.....		5.3	62.6	7.9	3.5	11.7	25.8	48.4	28.3	34.2	26.5	3.1	1.9	5.5	4.9	11.7	22.7	15.0	15.0	1.0	0.3	15.8	13.7	17.8	30.4	17.8	30.4	17.8	30.4
Sink in 4.20 sp. g....		60.9	56.3	39.1	31.3	62.1	62.4	62.4	60.7	85.3	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100
- 325 mesh†.....		100	100	100	100	43.8	45.1	40.9	31.5	26.4	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100
						45.0	48.0	42.2	33.3	28.5	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100

* Estimated for lack of analysis.

† Not subjected to sink-and-float tests.

SINK-AND-FLOAT FOR DISSECTION OF CRUSHED ORES

The method employed is what has been known as "float-and-sink," but it is better called "sink-and-float," to remove the expression as far as possible from confusion with "flotation." In metalliferous work the tools for "ore testing" have been too limited. The screen analysis has always been available. After that came handsorting and microscope; then the "machinery testing" would begin. If the concentrating machine failed to make finished products the grinding machinery would again be set to work, and sometimes overworked.

In coal washing, this procedure will not do because undue grinding of coal is penalized directly by the trade. Thus the coal-washing engineer has used, "heavy solutions" to fractionate his coal into specific gravity increments so as to determine the minimum amount of grinding necessary to liberate the waste. His results were used as a guide in building the washeries and forecasting the nature of products therefrom. His only apparatus was a bucket of the solution and a skimmer to skim off the float from a solution of known density.

The "ore testing" of the coal-washing engineer has been simplified through the ease of obtaining solutions heavy enough to float coal; a saturated solution of zinc chloride would always suffice. But in metalliferous work obtaining heavy solutions is not so simple because the lightest gangue is as heavy as the heaviest material in coal. Liquids of greater density than 2.6 have to be obtained. They must be transparent, non-viscous, non-corrosive and have a low melting point.

The method depends almost entirely upon specific gravity of the grains; porosity might give misleading results. The shape of the grain, which is important in gravity concentration, also has to be determined by other means. Obviously, after separating an ore into screened sizes and "sinking-and-floating" each of these to make various specific gravity increments, and having free access to chemical analyses and the microscope, one may determine at what size the ore is sufficiently crushed. Only a coincidence of specific gravities could interfere. The method allows one to get really into the heart of the problem. For example, if a certain fraction shows by microscope or chemical analysis a troublesome amount of locked mineral this product may be crushed for further fractionation until the required amount of crushing has been determined.

Heavy solutions are often of great help to the chemist when quick and approximate results are required. When the material is not finer than about 150 mesh a separation can be made in a very few minutes. Finer material may be separated by allowing it to stand a few hours. For a long time geologists have used heavy solutions to isolate minerals for study and identification, but their solutions were of small volume because they were very expensive.

SELECTION OF HEAVY SOLUTIONS

One of the most easily obtained heavy solutions for the metallurgist is acetylene tetrabromide with density of 2.95. It is miscible with carbon tetrachloride in all proportions so that a range of density from 2.95 to 1.6 may be had. Acetylene tetrabromide is not, as yet, found in the general market but it may be bought from the Dow Chemical Co., Midland, Mich., for about \$1.50 per pound.

For a solution of greater density than the above mentioned, stannic bromide may be used, specific gravity 3.34. It melts at 30° C. A chemist can prepare it at a cost of about 75 c. per lb. It can be diluted with acetylene tetrabromide for densities between 3.34 and 2.95. Antimony tribromide, which was used in the investigation herein described, has the same general deportment and an even greater density, 3.7, but as a more congenial reagent was desirable, later on thallium malonate-formate was used for a density of 4.2. This is a very acceptable liquid.

SPECIFIC GRAVITY FRACTIONS FROM RESPECTIVE SIZES

Table 2 is divided into 10 sections, one for each size from 14 mesh to 325 mesh. Five fractions were made from each size. The first section of the table exhibits the coarsest size, 14 to 20 mesh. Throughout this section the marked similarity of the respective ores is outstanding. The float on 2.9 averages only about 23 per cent. weight and its tenor in iron is about 6 per cent. The fraction between 2.9 and 3.34 is really too rich in iron to be discarded—about 20 per cent. Thus there is no opportunity to discard a tailing coarser than 20 mesh. On looking at the fraction which sinks in 4.2 we see that it is insignificant in quantity—about 5 per cent. weight, and the grade of it is below 60 per cent. iron; whereas, it would approach 70 per cent. iron if clean free hematite predominated. From this size, no matter how ingenious the mechanical devices might be, it would not be desirable to make either a concentrate or a tailing. Further difficulties are encountered with the lime because it stays with the light gangue when it is desirable to have it with the iron. The characteristics of the succeeding sizes show only slight changes as compared to the one described so that only a moderately more favorable condition is found in the 48-to-65-mesh products. The lighter products of this size could not consistently be thrown away and the heaviest is yet below 60 per cent. iron content.

The last section of the table deals with 270-to-325-mesh material. This size might be expected to contain only free gangue and free hematite, but not so. The 2.90 to 3.34 solutions segregated a product from each ore that contained about 20 per cent. iron and the sinks in 4.20 were only a little above 60 per cent. in grade.

The heaviest grains are gray and the lightest grains with their veneer of hematite on a siliceous background are red; the siliceous grains are like a used "streak plate." This would be a guide to the operator if tables could be employed. From the beginning to the end of Table 2 the "insoluble" in the heaviest sinks is too high to permit a cheerful outlook on concentration. In another way the table lends encouragement because it indicates that a process acceptable for one of the ores would be acceptable for all.

SIZING ANALYSES OF THE INSOLUBLE PART OF THE ORES

The insoluble constituents are the objectionable part of an iron ore. The object of crushing is to liberate them. The degree of liberation has been shown in Table 2. The mesh of the insoluble grains is given in Table 3. About 1 lb. of each of the five samples, crushed through 14 mesh, was digested with hydrochloric acid until all of the hematite was

TABLE 3.—*Sizing Analysis of "Insolubles" from Respective 14-mesh Ore Samples after Digestion in Hydrochloric Acid*

Mesh	Weights, Per Cent.				
	Ore No. 1	Ore No. 2	Ore No. 3	Ore No. 4	Ore No. 5
— 14 + 20	6.43	8.88	9.48	4.43	8.45
— 20 + 28	12.18	15.90	1 16.25	8.04	12.63
— 28 + 35	11.30	17.98	8.08	6.29	9.73
— 35 + 48	12.18	14.88	13.67	6.88	6.40
— 48 + 65	8.87	6.99	5.48	4.66	5.12
— 65 + 100	7.48	6.65	5.62	5.83	10.01
— 100 + 150	4.78	4.42	3.25	4.66	7.62
— 150 + 200	2.26	2.71	1.63	2.91	5.17
— 200 + 270	3.48	3.88	2.51	6.18	5.67
— 270 + 325	1.48	.38	1.01	3.15	2.50
— 325	29.56	17.53	23.02	46.62	26.70
	100.00	100.00	100.00	100.00	100.00

dissolved. Nothing but insoluble matter remained as a residue. This residue was wet-screened and the results are shown in Table 3. Visual examination of the residues shows that all of the ores are oölitic. Some of the round water-worn sand grains were too large to pass the 20-mesh sieve. The residue from each sample showed about 6 per cent. weight on 20 mesh. The oölitic consist of quartz, volcanic ash, feldspars of the albite-anorthite series and other sea-beach grains not identified. A constituent of another sort was abundant and it is the most important of all. It was very soft and white, and although a few of the grains remained on the coarse sieves, most of them were so soft that upon gentle rubbing, they readily passed through the finest sieve and swelled the per

cent. weight of the -325 mesh. The substance is claylike and will be called clay. In discussing the screen analysis in Table 1 the absence of free clay was commented upon. But clay can be identified in abundance after the hematite is dissolved. It seems to have been an integral part of the hematite so that it was liberated when the hematite went into solution. Such a theory would account for the low grade of the heaviest sinks shown in Table 2.

Table 3 further shows by the smallness of the per cent. weight between 100 mesh and 325 mesh that most of the finest sea sand was absent in the beginning, hence the conclusion that if the ores are to be crushed finer than 100 mesh most of the energy of grinding will be spent on further reduction of siliceous grains already reduced by grinding to 100 mesh. No evidence is available that grinding liberates the clay from the hematite. Fine grinding will, of course, remove some of the veneer of hematite from the old surface of the broken gangue.

DEGREE OF FINENESS OF INSOLUBLE IN RESPECTIVE SPECIFIC GRAVITY FRACTIONS OF A 14-TO-20-MESH SAMPLE

Further evidence on the distribution of clay and hard siliceous grains is in Table 4. Ore No. 2 was selected as representative. Sink-and-float was applied to the part between 14 and 20 mesh. The respective fractions were digested in hydrochloric acid and the sizing analyses of the residues are given. The residue from the float on 2.70 was coarse and oölitic; 88 per cent. was coarser than 20 mesh. More fines appeared in

TABLE 4.—*Sizing Analysis of Insoluble Matter in Respective Specific Gravity Products Made from 14 to 20-mesh Part of Ore No. 2*

Weights, Per Cent.						
Mesh	Residues from Specific Gravity Products					
	Float 2.70	2.70 to 2.80	2.80 to 2.90	2.90 to 3.10	3.10 to 3.34	Sink 3.34
— 14 + 20	88.00	80.78	65.30	37.42	10.67	.36
— 20 + 28	11.22	14.87	26.47	38.77	32.31	8.60
— 28 + 35	.39	1.42	3.33	8.37	18.33	17.95
— 35 + 48	{	.87	1.42	4.16	10.52	15.07
— 48 + 65		.32	.63	1.46	4.21	5.96
— 65 + 100		{	.63	1.56	3.31	4.42
— 100 + 150			.47	1.04	1.80	2.09
— 150 + 200			.87	.32	.73	1.20
— 200 + 270	.32		.83	1.12	1.84	
— 270 + 325	{	.16	.36	.68	1.11	
— 325		.87	.95	5.30	15.85	41.31
	<hr/>	<hr/>	<hr/>	<hr/>	<hr/>	<hr/>
	100.00	100.00	100.00	100.00	100.00	100.00

the residues of successive heavier fractions until in the residue of the 3.34 sink about 40 per cent. was claylike and was easily put through the 325-mesh screen by wet screening. Here is visual proof of the segregation of clay in the richest hematite and of siliceous grains in the leaner portion.

The ocular evidence of clay in the heavy grains is confirmed by chemical evidence in Table 5.

CHEMICAL ANALYSES OF RESIDUES

Table 5 deals further with insoluble constituents and shows by chemical analysis the amount of clay. The last column, called "difference," includes the oölitic grains of quartz, feldspars, etc. The first four lines deal with "sinks" in a solution with a density of 3.7. As 65 mesh is

TABLE 5.—*Character of "Insoluble" Incorporated in Richer Grains of Ore No. 5*

Mesh	Specific Gravity	"Insoluble" Assay, Per Cent.	Clay Assay, Per Cent.	Difference Assay, Per Cent.
— 20 + 28	Sink 3.70	14.2	8.5	5.7
— 28 + 35	Sink 3.70	13.2	7.4	5.8
— 35 + 48	Sink 3.70	13.4	8.9	4.5
— 48 + 65	Sink 3.70	12.6	9.4	3.2
— 65 + 100	Sink 3.34	15.4	9.5	5.9
— 100 + 150	Sink 3.34	13.2	9.9	3.3
— 150 + 200	Sink 3.34	15.0	9.3	5.7
— 200 + 270	Sink 3.34	14.8	9.1	5.7
— 270 + 325	Sink 3.34	15.5	9.3	6.2

approached all of the sizes show about the same amount of clay, but the "difference" column shows a slight decrease in the amount of grains of quartz and feldspars. This is evidence that grinding does not liberate clay and that by the time 65 mesh is reached the other contaminants are small in amount. That is, further grinding could do very little toward enriching the richest grains, although, of course, it would increase possible commercial extraction by liberating some of the hematite from the leaner grains. Therefore, grinding beyond 65 mesh helps primarily to increase the extraction and second to increase grade only so far as medium-grade grains are crushed and thus prevented from carrying gangue into the concentrate. The second section of Table 5 deals with the respective sizes from 65 to 325 mesh. In it there is no sign of diminishing insoluble in passing from 65 mesh to 325 mesh. This is not a material variance with Table 2, where a very slight increase in richness is shown when reading downward through the finest sizes.

The nature of the residue remaining after dissolving the hematite out of the 4.20 sinks bears out the theory that clay permeates the iron grains; the residual grains are of about the same size as before lixiviation, and when broken yield a claylike dust or shells of sundry shapes. This is brought out by Fig. 1.



FIG. 1.—PHOTOMICROGRAPH OF RESIDUE AFTER LEACHING IN HCL GRAINS HEAVIER THAN 4.2.

The insoluble shells are of about same size as original grains—48 to 68 mesh—although more than 85 per cent. of the substance went into solution.

LIBERATION BY GRINDING

The bane of the coal washery is inherent ash, and the pest of the washery that attempts to beneficiate the red ores here under consideration will be inherent “insoluble,” a part of which is clay. A glance at the iron content of the heaviest sinks, turning back to Table 2, shows that if they could be segregated a concentrate of 60 per cent. grade is all that could be expected. The possible extraction will depend upon the amount of grinding.

The data in Table 6 have been compiled to show the “ideal” extraction from size to size. The heaviest sink in each size is the assumed concentrate. Ore No. 1 would yield only 7.9 per cent. of its iron in the coarsest, and 50.8 per cent. from 48 to 65 mesh; a satisfactorily high extraction could not be made from 48-mesh material and lean grains

would contaminate the concentrate. The sizes from 65 to 325 mesh show improved extractions from the coarsest to the finest, and the assumed concentrates which are the sinks in 4.20 assay a little more than 60 per cent. iron. The inherent clay and silica amount to about 1 part to 8 parts of iron.

TABLE 6.—*Ideal Extraction of Iron in Concentrates Made from the Respective Sizes*

Mesh	Specific Gravity	Iron, Per Cent.				
		Ore No. 1	Ore No. 2	Ore No. 3	Ore No. 4	Ore No. 5
— 14 + 20	Sink 4.2	7.9	9.3	11.2	12.2	2.1
— 20 + 28	Sink 4.2	13.4	9.1	23.9	17.7	27.3
— 28 + 35	Sink 4.2	31.5	25.5	35.5	24.9	27.0
— 35 + 48	Sink 4.2	40.2	42.8	39.8	28.9	32.0
— 48 + 65	Sink 4.2	50.8	56.5	39.2	38.9	34.4
— 65 + 100	Sink 4.2	55.5		62.2	53.6	49.2
— 100 + 150	Sink 4.2	70.0		69.9	67.2	59.9
— 150 + 200	Sink 4.2	76.0		73.5	71.0	63.7
— 200 + 270	Sink 4.2	85.3		82.6	78.4	67.3
— 270 + 325	Sink 4.2	86.4		85.5	77.8	71.6

If the coarse sizes, say the 20 mesh, were reduced to the finest size they would likely give products similar to those described.

What has been said in the foregoing about degree of liberation after grinding is summed up in Figs. 2 and 3. In Fig. 2 is shown by diagram the specific gravity products on the respective screen sizes. The per cent. weight of the respective screen sizes, *i.e.*, the sizing test, is laid off on the horizontal lines, and the per cent. weight of the specific gravity increments is shown on the vertical scale. The ratio of a given small rectangle to the large rectangle shows the relation that a given increment of a given size bears to the whole. For example, the small rectangle in the upper left-hand corner, Fig. 2 ore No. 1, pictures the percentage weight of the plus-20-mesh part that has a specific gravity less than 2.90. Reading horizontally along this zone shows how this increment varies on passing to the smaller sizes. On going back to the upper left-hand corner and reading downward one sees the distribution of the increments obtained from the 14-to-20-mesh size. Reading along a horizontal zone shows the amount of a given increment through all the sizes. Reading along a vertical zone shows the distribution of the increments in a given size. An examination of the right-hand side shows that liberation is not complete on sizes as small as 270 mesh.

Facts of further interest are contained in Fig. 3 where the distribution of the iron is shown. The large portion of the iron contained in incre-

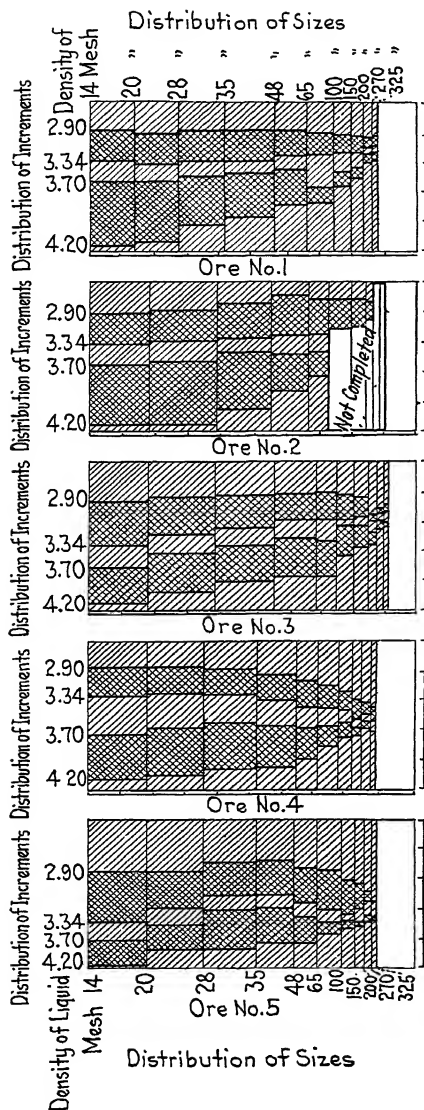


FIG. 2

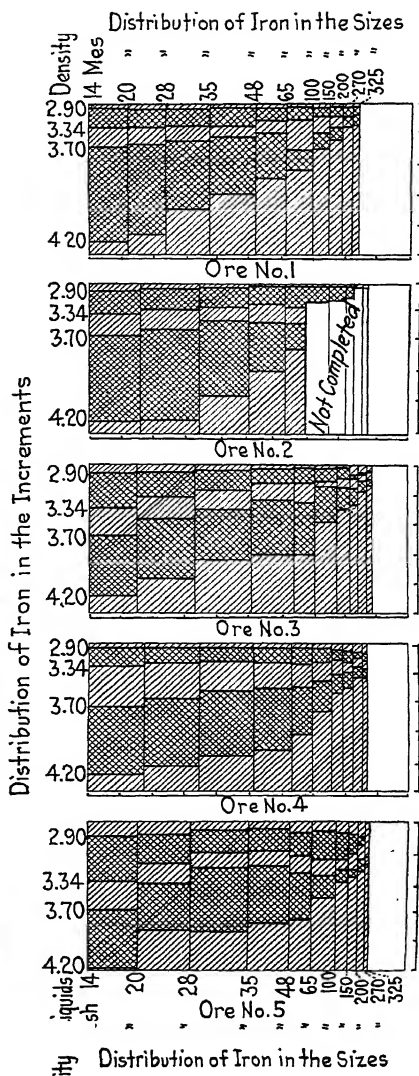


FIG. 3.

FIG. 2.—SPECIFIC GRAVITY PRODUCTS BY SINK-AND-FLOAT METHODS.

FIG. 3.—DISTRIBUTION OF IRON IN THE SPECIFIC GRAVITY INCREMENTS AND THE SIZES.

ments of intermediate densities makes concentration seem difficult. Only in the finest sizes is the iron abundantly segregated in the heavy and rich grains. A glance at the diagrams shows that these five ores have very much the same properties. It is not at all unlikely that during the life of any one of the mines represented by these samples the nature of the ore would vary as much as the samples herein delineated. Furthermore, anyone who is interested in the beneficiation of these ores should extend the investigation by obtaining other samples to ensure a more exact representation of the mines' output.

High extraction and high grade of concentrate can not be obtained without grinding to a size comparable with flotation or cyanide practice. If grade of concentrate be sacrificed the grinding need not be so severe.

APPLICATION OF MACHINES

The only feasible way to concentrate these ores seems to be by the use of roasting furnaces to make the hematite magnetic, and magnetic log washers for concentrators. This work has been done in a 300-lb.-per-hr. miniature plant at the Southern Experiment Station of the U. S. Bureau of Mines by Oscar Lee, B. W. Gandrud and F. D. DeVaney. Very fine grinding is required. The little plant is well organized and yields very high extraction and a fair grade of concentrate. The process from grinding the ore to sintering the concentrate is clearly demonstrated. It will be described in a forthcoming report by the above-mentioned investigators.

ACKNOWLEDGMENTS

The fractionations and calculations were performed by H. E. Messmore. Much credit is due him for his painstaking work. C. O. Anderson assisted in the preparation of the manuscript.

DISCUSSION

H. D. PALLISTER, University, Ala. (written discussion).—Mr. Coghill is to be congratulated on his careful analysis of the condition and degree of separation of the minerals in the low-grade red hematite ores of the Birmingham District of Alabama. During the past thirty or forty years many others have tried to solve the problem of separation of these ores, both on an experimental and a commercial scale and with varying degrees of success. Mr. Coghill has gone into the matter, in its most elemental aspect, that of the minute grains of minerals and their relation after grinding. This is where the separation should start.

At first sight the problem seems simple, but when the ores have been carefully studied, the close interlocking of hematite and silica becomes apparent. A small amount of high-grade concentrate can be readily obtained but the small percentage of recovery of the iron content of the original ore renders the problem one which does not appeal as a commercial venture. The result is that the experimenters have devoted their efforts to the problem of refining their crushing processes with the aim of developing one which will set the hematite free from the silica without undue sliming of the

hematite content. When this product is obtained, the problem of a cheap means of separation is still to be applied. Specific gravity and magnetic processes of separation or combinations of the above have been tried with various degrees of success, dependent on conditions.

Such work as that done by Mr. Coghill will facilitate the ultimate solution of the problem on a commercial basis. The U. S. Bureau of Mines, through the Southern Experiment Station, in coöperation with the State Mine Experiment Station of Alabama and with the assistance of the Minnesota State Mine Experiment Station, has been earnestly engaged on this problem for a number of years and has solved some phases of it in a satisfactory manner.

R. M. RAYMOND, New York, N. Y.—Mr. Singewald, has any actual work in experiments been done by any of the operators?

J. T. SINGEWALD, JR., Baltimore, Md.—I do not think the operators have done anything. The Bureau of Mines is experimenting with commercial machines.

E. F. BURCHARD, Washington, D. C.—I suggested to a mining man in Birmingham that he comment on the Coghill and Singewald papers³ and he has written me as follows:

"Coghill's paper seems to indicate that very fine grinding is necessary, which is in line with the Bureau of Mines experiments at Tuscaloosa, whereas I recall they ground the material to below 200 mesh and produced a concentrate of 55 per cent. iron. Singewald seems to recommend coarser crushing to about 20 mesh and production of a lower grade concentrate. Singewald is correct in stating that heretofore the economic aspect has been entirely neglected, and from what data were given out at Tuscaloosa I feel that this side of the matter was not very closely considered there.

"In regard to the concentration of these ores, I do not feel that the bottom bench ore is so important as the siliceous ore in the Big Seam in the north part of the field. As you know, the bottom bench has only important thickness in the north part of the field in the Ishkooda vicinity, and this group of mines is now working close to 12 ft. of ore, and in places thicker. Ore below this on the lower bench is extremely low grade and the cost of mining would, I feel, be higher or at least as high as the average ore now mined. There is much that we do not know about the bottom bench of the Big Seam as to its extent.

"In the vicinity of Shannon, this ore is all being mined with recovery greater in the bottom bench than the top bench, and if similar conditions hold over this general vicinity it will all be mined under present conditions.

"In regard to the value of a concentrate as described by Singewald, I do not believe a 45 per cent. iron and 15 per cent. silica would exceed, perhaps scarcely equal, the unit value of an average brown ore from 45 to 50 per cent. in iron, as any furnace man feels that a brown ore of good grade is easier to smelt than the red iron ore, and has some special advantages when used as a mixture with red ore to improve the fluxing qualities.

"From what I have seen of sintered products, I also feel that there is much uncertainty as to the cost and physical qualities that can be produced by sintering. The percentage of fines and improperly sintered soft material is high in such products. In general, I feel that there is a very considerable distance yet to go in the way of concentration on a commercial scale.

"These are only impressions I have received from hasty reading of the paper.

"The Bureau at Tuscaloosa gave a general opinion, based on costs which I felt were too low, that a sintered concentrate could be produced between 50 and 55 per cent. iron at \$5 per ton, which would be 9 or 10 c. per unit as compared with the 5 or 6 c. at present prices."

³ J. T. Singewald, Jr.: *Op. cit.*

D. A. LYON, Washington, D. C.—In taking up this work, there was no intention of developing a process. The object of the work was to secure fundamental data that will enable the industry to get ready for the time when it will be necessary to use these ores. We realize that this work is something for the future—five, ten or twenty years. We do not know when it will come.

W. M. GRANT, Birmingham, Ala.—The published data of the experiments made by the Bureau of Mines on Minnesota ores show a concentrate can be made containing about 85 per cent. metallic iron by direct reduction with subsequent magnetic concentration. I wonder whether any similar determination has been made on this hard ore in Birmingham.

T. L. JOSEPH, Minneapolis, Minn.—I know of no direct experiments in which attempts have been made to metallize the ores in the Southern District, but I feel certain that little difficulty would be encountered in mixing the Southern ores with coal and metallizing them just as has been done with the Minnesota ores. However, that is only a small part of the entire program that would be necessary in order to utilize the ores in this way.

The matter of producing steel by what is commonly known as direct processes has a very uncertain status at the present time, but it is being investigated and may possibly provide a means of treating those ores; although the amount of slag that would be involved in melting the sponge iron after the ore has been metallized would be much greater than would be true of the Northern ores. This would involve an iron loss due to the fact that there would be some iron in the slag produced upon melting the sponge iron. On the whole, I do not think it looks very favorable, at least at the present time.

O. C. RALSTON, Berkeley, Calif.—In answer to the question that the previous speaker raised, I see no value in metallizing iron first in order to get a separation. As a rule, metallizing causes partial melting or sintering of portions of the ores and locking of grains together so it is more difficult to separate them after the heat treatment than before. Of course, there are isolated cases where various kinds of ores tend to split along the planes of contact of diverse minerals after previous heat treatment. In general, the increased ease of crushing and cleanliness of breakage are worth consideration on some finely disseminated ores, but I think that applies more to other types of ores than to iron.

In Mr. Coghill's paper, he refers to a paper⁴ on heavy liquids. I am responsible for the work of a laboratory in Berkeley, which turned out, at request, a piece of work on heavy liquids which geologists have used for years, and if you will consult textbooks on geology or mineralogy, you will find lists of heavy liquids, many of which are not well adapted to such work. The paper reporting the work is not published yet, but in it we have given a good estimate on the best type of heavy liquids. Coghill uses acetylene tetrabromide; this is an ideal liquid—an organic material—as it is easily removed from the particles and does not attack the material. For heavier specific gravities, only one other liquid has to be considered, thallium malonate-formate, which can be washed from the particles with water. The heavy liquid separation gives you your yardstick on the mechanical concentration of ores.

S. M. MARSHALL, New York, N. Y. (written discussion).—The use of moderately heavy liquids for the investigation of minerals requiring beneficiation is not new; they have been employed many years in the study of coal and of some of the lighter minerals for which the lower density liquids are suitable. Mr. Singewald has described⁵ the much earlier use of heavy liquids with the iron ores of Alabama and both in this country and abroad they have been used for similar separations.

⁴ J. D. Sullivan: *Op. cit.* (see p. 147).

⁵ J. T. Singewald, Jr.: *Op. cit.*

The new feature which Mr. Coghill presents is the use of liquids of greater specific gravities than have been employed heretofore, which enable ores to be divided over a much wider range than previously has been possible. This opens opportunities for the more thorough investigation of iron and other ores and provides a tool for experimenters in ore dressing similar to the one that has been used so successfully in the study of coal-washing problems.

Mr. Coghill is primarily interested in determining the extent to which the mineral values are released from a crude Alabama ore by fine grinding, and to learn the maximum size at which an economical recovery can be obtained. The diagrams given at the end of his paper illustrate the results and show the extent to which the iron oxide is released from the gangue by successively finer grinding. For this purpose these diagrams are excellent because they show the point at which there is a sharp improvement in the separation.

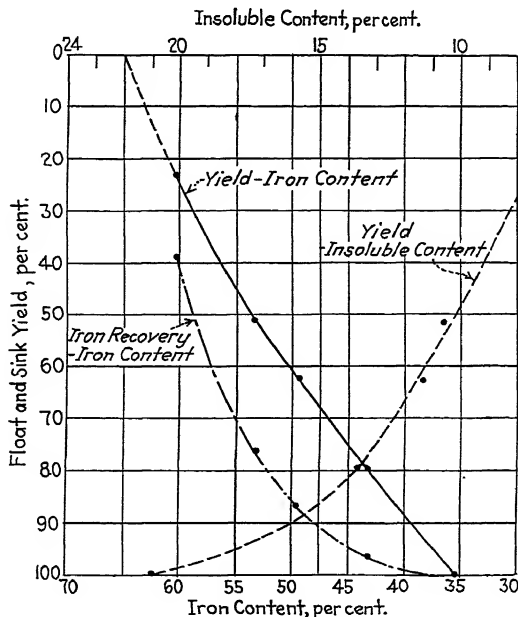


FIG. 4.—ALABAMA RED IRON ORE FLOAT-AND-SINK YIELD CURVES BASED ON DATA FROM PAPER BY W. H. COGHILL. CURVES ARE COMPOSITES OF ORE NO. 1, COMBINING ALL SCREEN SIZES EXCEPT -325 MESH.

For practical operations another form of curve is valuable and this is one showing the relation between the iron content of the concentrate and the float-and-sink weight yield. It is customary to use a curve of this type in coal washing and it is of great value because it shows at a glance the maximum possible recovery of a finished product with a given analysis from a low-grade raw material. Curves of this kind are used by the Northwest Experiment Station of the Bureau of Mines and other laboratories for coal washing.

Fig. 4 shows a relation for the No. 1 ore in Mr. Coghill's paper. This is a cumulative or composite curve for all of the screen sizes except -325 mesh; this finest size was not separated by Mr. Coghill. The curves can be taken to represent the average ore safely enough because the -325 mesh size is higher than the average of the coarse

sizes. As some of these fines will be recovered in the washing operation, they will raise rather than lower the grade of the washed ore, consequently any deductions made from the coarse sizes will be conservative.

On the plate are three curves; the principal one is the yield iron content curve which shows the relation between the percentage of iron in the sink ore and the percentage of weight recovery of the concentrate at that analysis. The curve passes through five points which represent the cumulative results at the several specific gravities used by Mr. Coghill. For instance, the raw ore contains 35.56 per cent. iron and this represents 100 per cent. of the weight. When this is separated on 2.90 specific gravity, the portion that sinks amounts to 79.86 per cent. of the weight and it has an iron content of 43.03 per cent. When separated upon 3.34 specific gravity, the portion that sinks is 62.41 per cent. of the raw ore and its iron content is 49.47 per cent. The other two points represent the sink material in 3.70 and 4.20 specific gravity, respectively.

The dashed curve on the plate shows the relation between the percentage yield and the insoluble content of the concentrated ore. The points on this curve correspond to those on the yield iron-content curve and are to be read on the yield ordinates and the insoluble percentage along the abscissa.

The third curve, which is drawn with dots and dashes, shows the percentage of the total iron in the raw ore which is recovered at any iron content of the concentrate. This curve is read on the percentage scale at the left and the iron percentage scale at the bottom. There are five points for these latter curves, as in the first case. The use of these three curves will be made more apparent by taking a specific case.

If it be assumed with an ore of the type discussed by Mr. Coghill that the iron content of the beneficiated product should not be less than 50 per cent., the yield iron-content curve indicates that the weight yield of this grade will be 61 per cent. With this yield the insoluble content will be 11.4 per cent. and the recovery of the metallic content of the ore 85.6 per cent. While the insoluble content of such an ore will appear somewhat high to those accustomed to Lake Superior ores, the presence of lime in the Alabama ores will offset a portion of the insoluble because a 50 per cent. ore will carry about 5.40 per cent. lime. Consequently the insoluble, which must be fluxed by additional stone, will be only about 6 per cent.

It must be kept in mind that these float-and-sink yield curves show the maximum recovery that can be obtained in a washing plant employing hydraulic separation of any type, unless the ore be given some further mechanical treatment and be in a different mechanical condition than was the case when the laboratory separations were made. Actual operations will not give such large recoveries. The ratio between the practical plant yield and the float-and-sink yield is usually termed the efficiency of the washing operation. Probably at 50 per cent. iron content the efficiency would not exceed 85 per cent., which means that an iron ore containing 50 per cent. iron and 11.4 per cent. insoluble would show a practical recovery of 51.8 per cent. by weight with a metallic recovery of 72.7 per cent.

The use of these heavy solutions for the float-and-sink separations of iron ores is so new that there are few data on different ores in the country and practically none comparing commercial operations with the laboratory results. It is desirable that the Bureau of Mines, in one or more of its experiment stations, should continue this work and increase its value by obtaining comparative data from jigs or other washing units to compare with the same ores separated on heavy solutions. The ratio between these two yields will give efficiency factors which will enable comparatively inexpensive and rapid laboratory tests to be converted into practical figures with reasonable assurance of accuracy.

The work which the Bureau of Mines is doing in its experiment stations on coal and metallic ores, and in the investigation of the scientific and practical relations in

pyro- and hydro-metallurgy is of exceptional value to the mineral industry of this country. The Bureau is able to pursue investigations which could not be undertaken by individual companies, with the added advantage that the results are available alike to large and small enterprises, and it is to be hoped that funds will be provided permitting the work to be continued and amplified.

R. GAHL, Berkeley, Calif. (written discussion).—This paper represents a very valuable contribution toward eliminating the personal equation from concentration tests, and it would seem to be possible by the application of such a combination of screens and heavy solutions to design a mill without recourse to tests with actual concentrating machines either full size or of miniature construction.

I am not prepared to say that we have arrived at a similar point so far as the crushing and grinding part of a concentrating plant is concerned. As a matter of fact, the rationale in the design of grinding machinery installations seems to me to be in a chaotic state as yet. Successful efforts are being made, however, particularly under the auspices of the Milling Committee of this Institute, to clear up some fundamentals in that field.

So far as the application of the flotation process is concerned, we are, naturally, not even so far advanced as in grinding questions and the selection of flotation agents for the separation of minerals still follows rule of thumb methods. Considerable research, will be required before Mr. Coghill or anyone else can write papers in the fields of grinding and flotation corresponding to the paper under discussion.

An Introduction to the Iron-chromium-nickel Alloys

BY EDGAR C. BAIN* AND WILLIAM E. GRIFFITHS,† LONG ISLAND CITY, N. Y.

(New York Meeting, February, 1927)

SUMMARY

THE results of an inquiry into the structural nature of some 70 iron alloys containing both nickel and chromium over a considerable range of concentration are briefly described in this paper. This study is a part of a comprehensive program of investigation into alloys involving chromium in progress at the Union Carbide and Carbon Research Laboratories. All the compositions studied contained carbon in amount comparable to commercial alloys.

The B Constituent

In addition to locating the limits of composition for the three principal types of structure (polyhedral ferrite, martensite and austenite) and their combinations, certain other features of the system were observed. A hitherto undescribed brittle constituent (designated here as the "*B* constituent") was discovered. It is very hard, non-magnetic and forms in two modes from chrome-ferrite by recrystallization. It is the stable low-temperature condition for the equal-proportion chromium-iron materials containing less than 10 per cent. nickel. Although it partakes of the nature of an inter-metallic compound, yet it may exist over a considerable range of composition. It is non-existent above a temperature zone, varying with composition, from 800° C. to 950° C., and heating above this critical temperature develops chrome ferrite persistent after average cooling rate; a few hours just below this temperature will develop the complicated brittle crystalline type.

The *B* constituent, when found in the grain boundaries of alloys of lower chromium content, contributes a serious brittleness removable, however, by heating and cooling rapid enough to prevent its re-formation. By suitable means of precipitation, the hard *B* constituent can be made to strengthen the already stiff alloy matrix in the manner of submicroscopic particles, as exemplified in duralumin or the Sykes iron-tungsten and iron-molybdenum alloys.

A possible explanation is offered for the familiar brittleness developed in chrome iron at temperatures above about 475° C. and removable by

* Research metallurgist, Union Carbide and Carbon Research Laboratories, Inc.

† Union Carbide and Carbon Research Laboratories, Inc.

heating to a higher temperature followed by rapid cooling. This procedure is described by Becket and MacQuigg,¹ who have effected an increase in malleability and ductility in iron-chromium alloys by rapid cooling through the temperature range 600° C. to 400° C., permitting thereby the manufacture of deep-drawn or highly worked articles such as wire.

NATURE OF IRON-CHROMIUM-NICKEL ALLOYS

A first approximation of the nature of the ternary alloys of any three metals can usually be inferred from an inspection of the three binaries. If the three metals, *A*, *B*, and *C*, form a set of binaries *AB*, *BC* and *AC*, which are wholly solid-solution alloys, then the ternary will likely be composed wholly of solid solutions. At first glance, Fe, Cr and Ni form such a system. However, the situation is complicated by the fact that chromium and nickel form two series of solid solutions, chromium in nickel and nickel in chromium. This fact is revealed because the metals show different types of atomic arrangement. Chromium, like alpha iron, exists in the body-centered cubic arrangement of atoms, whereas nickel and gamma iron exist in the face-centered cubic space lattice.

Obviously, no two metals could develop a single homogeneous series of solid solutions unless they were of the same identical crystal type. Thus we can conceive of a continuous series of solid-solution alloys from pure alpha (or delta) iron to pure chromium or from pure gamma iron to pure nickel, and this is precisely what exists. Hence, first of all the ternary diagram is certain to contain two solid solutions—one embracing gamma iron and nickel with some chromium in solution and the other embracing alpha (or delta) iron and chromium and containing some nickel. There will be the two-phase zone between these single solutions types. Possibly there might be the prediction of martensitelike structures after cooling from temperatures of gamma iron stability, and there certainly would be such anticipation if carbon were considered as a universally present fourth element.

This theoretical inspection of the possibilities of structures will be seen to be remarkably accurate and would constitute almost the whole of the results were it not for the development of a new phase, both in pure iron-chromium and in alloys containing some nickel, after prolonged annealing. This state of affairs could scarcely be predicted although in the alloys of copper and gold there is analogous behavior.

METHODS OF STUDY

The experimental work was directed to obtain the most detailed knowledge of the nature of the alloys in the least time and with the least

¹ F. M. Becket and C. E. MacQuigg: U. S. Patent 1454464.

effort. The nickel-rich alloys have been sufficiently well studied to indicate the fact that they are almost certainly solid solutions of the nickel solvent type, hence this range of composition could be eliminated from the investigation. At first, only compositions in 5 per cent. steps were selected for melting; later it was found advisable to add a few compositions under 5 per cent. nickel. The black circles in Fig. 1 show the range covered in this research.

Six-pound heats were melted in a high-frequency furnace in magnesia-lined Dixon crucibles. Armco iron and low-carbon ferro-chromium with shot nickel constituted the charge. To keep as high purity as possible, no deoxidizer was used. However, in one way, the heats were rather unsatisfactory for they contained more carbon than was desirable, which content was not uniform, tending to increase with the chromium content.

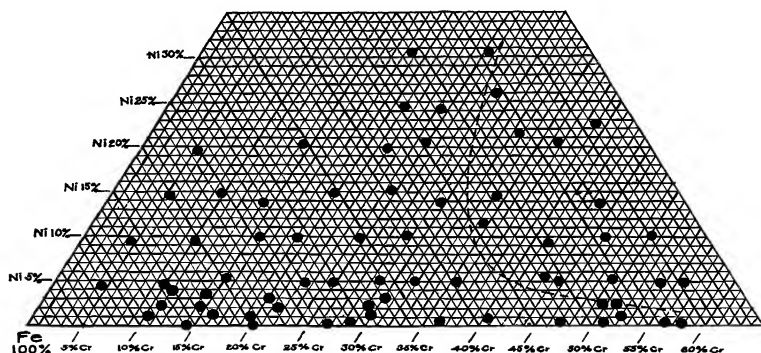


FIG. 1.—DISTRIBUTION OF ANALYSES OF METALS STUDIED.
Dotted line indicates zone of difficultly forgeable alloys.

All the alloys were forgeable although the ingots containing over 35 per cent. Cr and 5 per cent. Ni were most difficult to forge. The temperature range for successful deformation was very narrow and very high in this range of composition.

After the ingots had been forged into $\frac{5}{8}$ -in. square bars the metal was annealed for a few days in a large furnace with temperature falling gradually from 1000° C. over 4 days. This anneal would be as thorough as an average commercial anneal, but to find out more of the nature of the metal at equilibrium at the lowest possible temperature, a second annealing treatment was given small samples in which the temperature was dropped from 875° C. by 25° C. steps every 12 hr. The added effect was quite marked, as will be seen.

Small cubes of the metal from each annealed bar were heated for 1 hr. at 900° C., 1050° C., 1200° C. and 1300° C., and then quenched in water.

The structures found by microscopic examination in these four sets of quenched specimens provide the basis for the attempt to show the condition of the metals at various elevated temperatures. Although the microscope contributed much of the information concerning the behavior of the alloys, yet whenever ambiguity existed, the X-ray crystallometer was used to settle questions of identity of phase. The actual state of some of the structures resembling martensite was inferred from Rockwell hardness measurements.

In many of these alloys the austenite may be only partly decomposed or, at the other extreme, grains of ferrite often retain martensitic markings; yet one might judge both to be ordinary martensite from the polished and etched microscopic section. A very strong magnet was a constant aid in evaluating the proportion of ferrite present. Considerable skill may be acquired in such estimation by constant checking against microscopic appearance and X-ray films.

RESULTS

Before classifying the alloys into their types it may be desirable to present in consolidated tabular form the constituents found in each composition as given (1) the very long anneal, (2) the moderate anneal, (3) a quench from 900° C. into water, (4) a quench from 1050° C., (5) a quench from 1200° C. and (6) a quench from 1300° C. In the interest of brevity some descriptive words are used with certain specific meanings not entirely with the same restrictions as in the field of ordinary steel metallurgy:

Definitions of Words as Used Here

- Austenite: Any non-magnetic, non-brittle, solid solution showing twinning and the face-centered cubic atomic arrangement.
- Martensite: Any magnetic, moderately hard material showing the characteristic acicular manifestations of ordinary martensite under the microscope.
- Chrome-ferrite: Any polyhedral-grained magnetic material having the body-centered cubic arrangement of atoms. When so designated it may show retained markings resembling martensite although not adequately hard.
- B* constituent: (a) Massive form—a clear-etching, corundum-hard, brittle, non-magnetic constituent, usually filled with faint cracks;
- (b) Troostitic form—dark-etching solid dispersion of sub-microscopic particles of true *B* constituent in various amounts of ferrite matrix. Weakly magnetic, hard and often cracked. An intermediate form in ferrite changing to *B* constituent by long-time heating at low temperature.
- Carbides: Particles unattacked by the etch that seem hard. Due to tendency of these alloys to develop austenite out of carbide particles and also to throw out tiny needles of austenite under circumstances which would, in steel, be expected to reject carbide, there is often some doubt as to just which particles are carbides and which austenite. However, it is believed that in general the classification here will be found accurate.

MICROSTRUCTURE AND DETAILS OF THE ALLOYS BY TYPES

J. R. Vilella has photographed representative specimens of the types of alloys found. As in the case of the iron-chromium alloys careful polishing and repolishing is often required. The etching medium used here is the same used for iron-chromium alloys already described—equal parts of aqua regia and glycerine. Except when otherwise designated, the magnification at which the photomicrographs were made is approximately 75 diameters.

The Wholly Austenitic Alloys

All the alloys that are very rich in nickel are always austenitic at all temperatures, merely solutions of iron and chromium in nickel. The alloys of this classification are as follows:

COMPOSITION OF THE WHOLLY AUSTENITIC ALLOYS

ALLOY No.	CHROMIUM, PER CENT.	NICKEL, PER CENT.	CARBON, PER CENT.	ALLOY No.	CHROMIUM, PER CENT.	NICKEL, PER CENT.	CARBON, PER CENT.
8	15.7	9.9	0.12	29	24.9	20.5	0.18
12	19.1	9.9	0.20	34	32.7	21.5	0.24
13	14.1	13.8	0.12	30	21.0	24.5	0.12
18	19.8	14.9	0.24	35	24.4	24.2	0.29
85	25.0	14.9	0.14	42	28.4	26.0	0.18
19	14.3	20.3	0.17	36	18.6	30.5	0.14
24	21.9	19.9	0.18	43	25.4	30.6	0.22

These compositions are without transformation; with rise in quenching temperature they exhibit only grain growth. They are very soft, becoming softer the higher the nickel content and harder with increase of chromium. A very long annealing treatment causes some precipitation of excess carbide in parallel planes within the grains, sometimes on two families of planes simultaneously, developing a mesh system on the polished section. (See Fig. 2.) In certain cases it appears at high magnification that these rejected plates may be laminated with ferrite when the carbon and nickel content is relatively low. This is distinct from the ordinary destruction of austenite which develops the chevron-like ferrite precipitation later described.

Whether these austenitic alloys are all truly stable at room temperature is somewhat doubtful, for the atomic sluggishness is so great in any event that they cease to readjust themselves atomically or to restore equilibrium below temperatures probably as high as 300° C. However, a method may be used to determine this fact with more certainty. If such specimens of austenite are severely deformed at room temperature, they may harden inordinately and become feebly magnetic in the manner of Hadfield manganese steel. If this happens, then obviously the austenite is not truly stable at room temperature and the atomic agitation of cold work suffices in a small way to develop the truly stable form of

ferrite. Certain specimens in the iron-chromium rich austenites showed this tendency clearly. It is suggested that a series of deformation experiments at various temperatures would determine the true transformation temperature of the alloys. Fig. 3 shows the structures for this group of alloys.

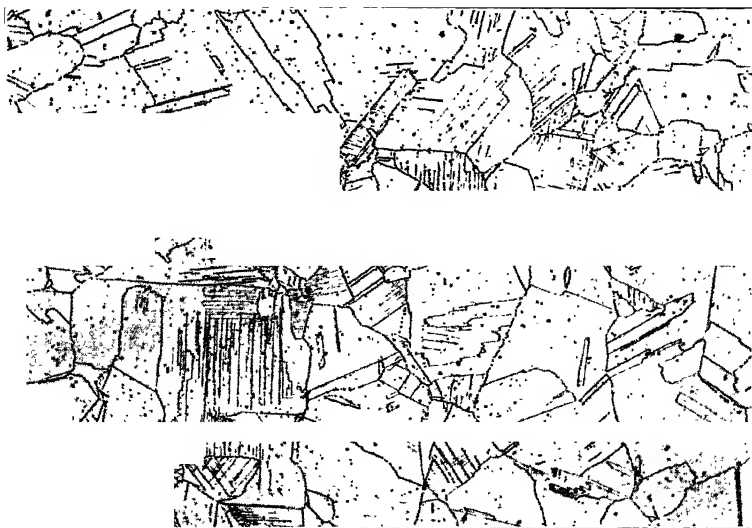


FIG. 2.—SLIGHT DECOMPOSITION OF AUSTENITE. $\times 500$.

Partly Transformed Austenite Alloys

COMPOSITION OF PARTLY TRANSFORMED AUSTENITE ALLOYS			
ALLOY No.	CHROMIUM, PER CENT.	NICKEL, PER CENT.	CARBON, PER CENT.
5	10.2	9.5	0.24
9		14.7	0.18

In this group are found those alloys that, while dominated by austenite and practically pure as quenched from just above 900°C ., still are not retained as austenite after anneal or as quenched from high temperature. Their alloy content is just too low to permit the austenite to persist. Such alloys would form a series of compositions represented by a fringe around the zone occupied by the first group described, the austenitic alloys. Fig. 4, alloy 5, is characteristic of the low-chromium type which tends toward martensite wherein the more drastic quench, probably by excessive thermal stress, forces martensitic transformation. Fig. 5, alloy 9, is characteristic of the higher-chromium type wherein the delta-iron influence (high-temperature ferrite development) causes chevronlike ferrite separation.

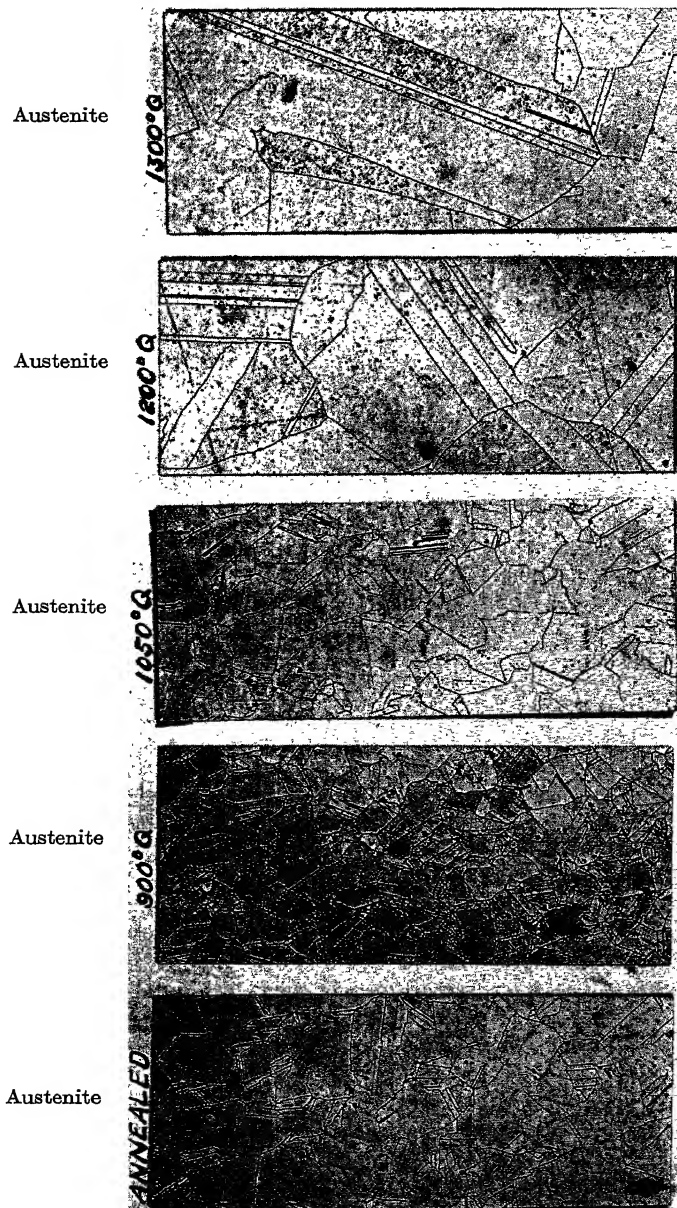
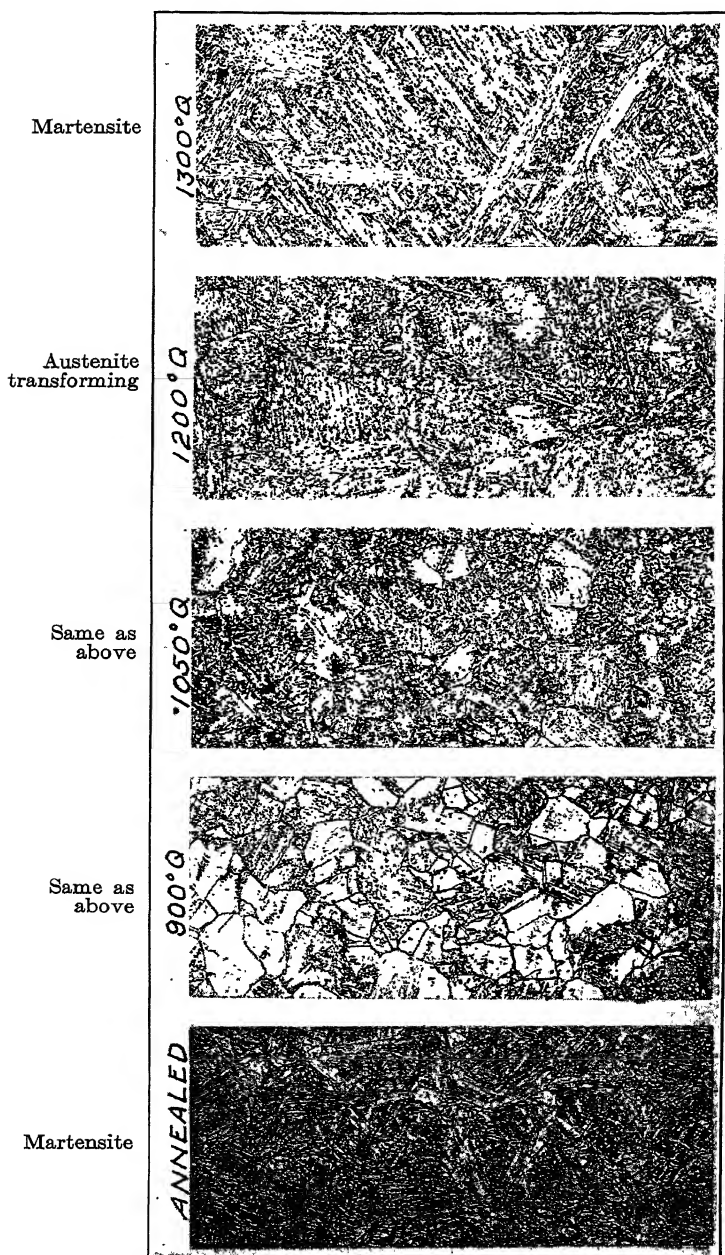


FIG. 3.—AUSTENITE STRUCTURES RESULTING FROM QUENCHES AT VARIOUS TEMPERATURES. $\times 75$.

FIG. 4.—SLIGHTLY UNSTABLE AUSTENITE ALLOYS OF LOW-ALLOY CONTENT. $\times 75$.

Austenite
transforming



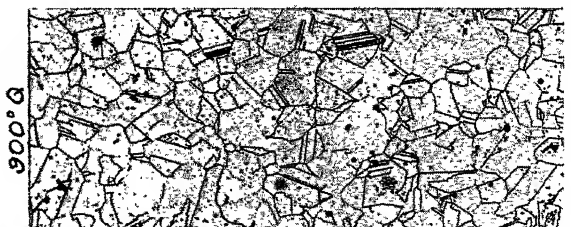
Same as
above



Same as
above



Austenite



Austenite
transforming

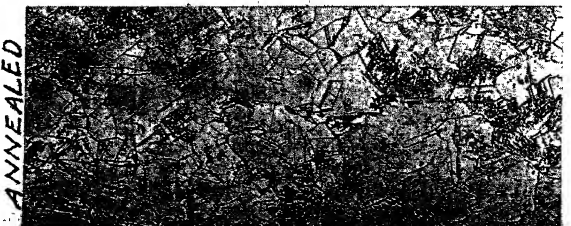


FIG. 5.—SLIGHTLY UNSTABLE AUSTENITE ALLOYS OF HIGH-ALLOY CONTENT. $\times 75$.

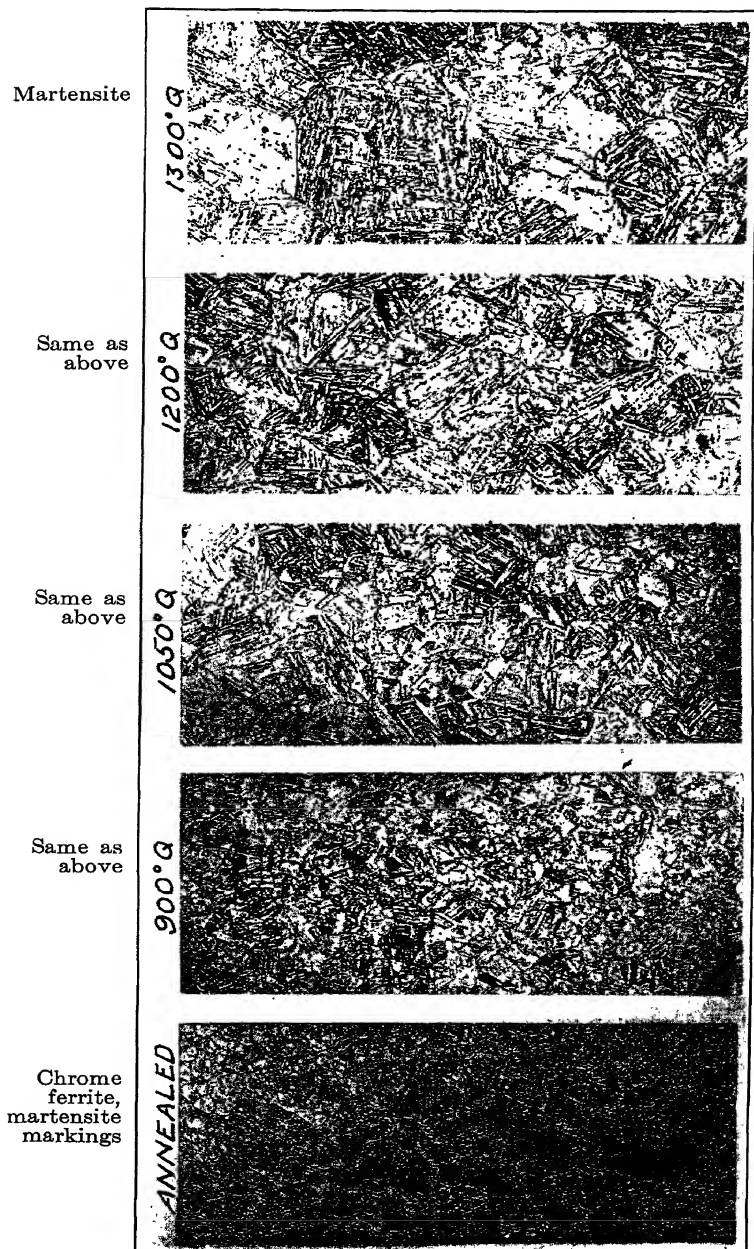
FIG. 6.—MARTENSITIC ALLOYS. $\times 75$.

TABLE 1.—*Constituents of Different Compositions of*

Alloy No.	Content			Extra Annealed	Annealed	900° C. Quench
	Cr, Per Cent.	Ni, Per Cent.	C, Per Cent.			
I. Alloys						
1	14.3	Trace	0.18	Chrome ferrite, few spheroidized carbides.	Chrome ferrite 80 per cent., pearlite 20 per cent.	Chrome ferrite 50 per cent., martensite 50 per cent.
3	20.2	Trace	0.12	Chrome ferrite, cleavage carbides.	Chrome ferrite, spheroidized carbides.	Chrome ferrite, pearlitic carbides.
6	26.8	0.3	0.24	Chrome ferrite, cleavage carbides.	Chrome ferrite, spheroidized carbides.	Same as annealed condition.
10	28.9	0.4	0.18	Chrome ferrite, spheroidized carbides.	Chrome ferrite, spheroidized carbides.	Same as annealed condition.
14	36.8	0.6	0.17	Chrome ferrite, spheroidized and cleavage carbides.	Chrome ferrite, spheroidized carbides.	Same as annealed condition.
25	47.3	0.3	0.44	B constituent, carbides.	Chrome ferrite, spheroidized carbides.	Same as annealed condition.
31	51.2	0.3	0.38	Chrome ferrite, carbides, few nuclei of B constituent.	Chrome ferrite, large spheroidized carbides.	Same as annealed condition.
37	56.6	0.3	0.42	Chrome ferrite, carbides or cleavage austenite.	Same as extra annealed condition.	Same as annealed condition.
44	58.0	0.18	0.36	Chrome ferrite, fine carbides or austenite.	Chrome ferrite, large spheroidized carbides.	Same as annealed condition.
II. Alloys containing about						
91	10.3	1.1	0.08	Chrome ferrite 50 per cent., badly formed pearlite 50 per cent.	Clear ferrite 10 per cent., martensite or pseudo-pearlite 90 per cent.	Martensite.
94	15.9	1.2	0.05	Chrome ferrite, fine; pearlitic carbides.	Chrome ferrite, fine 50 per cent.; pearlite 50 per cent.	Martensite 75 per cent., chrome ferrite 25 per cent.
97	19.5	1.1	0.08	Chrome ferrite, carbide in rings marking old austenite pools.	Chrome ferrite 50 per cent., pearlite 50 per cent.	Chrome ferrite 50 per cent., martensite 50 per cent.
100	29.8	1.2	0.09	Chrome ferrite, carbides.	Chrome ferrite, spheroidized carbides.	Chrome ferrite 80 per cent., austenite 20 per cent.
20	41.3	1.0	0.36	Chrome ferrite 85 per cent., cleavage austenite 15 per cent.	Chrome ferrite, spheroidized carbides.	Same as annealed condition.
103	52.1	1.0	0.31	B constituent, carbides or austenite, cracked.	B constituent, carbides.	Chrome ferrite, carbides, cracks largely healed.
III. Alloys containing about						
92	11.0	2.2	0.07	Chrome ferrite, martensitic markings 90 per cent., pearlite 10 per cent.	Martensite.	Martensite.

Chromium-nickel-iron Alloys after Different Treatments

1050° C. Quench	1200° C. Quench	1300° C. Quench	Alloy No.
very low in nickel			
Chrome ferrite 30 per cent., martensite 70 per cent.	Chrome ferrite 70 per cent., martensite 30 per cent., recrystallization in ferrite.	Chrome ferrite 70 per cent. in large grains with martensite, 30 per cent. in cleavages.	1
Chrome ferrite, coarse, 95 per cent.; martensite 5 per cent.	Chrome ferrite, coarse, recrystallization.	Chrome ferrite, very coarse; reprecipitated carbides, some coagulated in grain boundaries.	3
Same as annealed condition.	Reprecipitated carbides except near grain boundaries.	Chrome ferrite, carbides or austenite in cleavages.	6
Chrome ferrite, coarse, recrystallization, few carbides.	Chrome ferrite, coarse, recrystallization, carbide reprecipitation.	Chrome ferrite, cleavage carbides or austenite.	10
Same as annealed condition except incipient recrystallization.	Same as 1050° quench, except carbide reprecipitation.	Chrome ferrite, carbide reprecipitation except near grain boundaries.	14
Chrome ferrite, carbide reprecipitation.		Chrome ferrite, coarse, incipient fusion at grain boundaries.	25
Same as annealed condition.	Chrome ferrite, coarse, recrystallization.	Chrome ferrite, very coarse, incipient fusion.	31
Chrome ferrite, coarse, recrystallization; carbides in grain boundaries.	Chrome ferrite, recrystallization, reprecipitation of carbides.	Chrome ferrite, coarse; incipient fusion, carbides coagulated.	37
Same as annealed condition.	Chrome ferrite, reprecipitated carbides.	Chrome ferrite, incipient fusion, carbides or austenite in cleavage planes.	44
1 per cent. nickel			
Martensite, coarse.	Martensite, coarse.	Martensite, coarse, slight indication of incipient delta iron.	91
Same as 900° C. quench.	Chrome ferrite 50 per cent., austenite-martensite 50 per cent.	Chrome ferrite, coarse, 90 per cent.; austenite-martensite in cleavages 10 per cent.	94
Same as 900° C. quench.	Chrome ferrite 85 per cent., martensite 15 per cent.	Chrome ferrite, thin lines of martensite.	97
	Chrome ferrite, recrystallization, coalesced carbides.	Chrome ferrite, very coarse, carbide reprecipitation.	100
Chrome ferrite, recrystallization.	Chrome ferrite, recrystallization.	Chrome ferrite, coarse, incipient fusion, reprecipitated carbides.	20
Same as 900° C. quench.	Chrome ferrite, recrystallization, carbide reprecipitation.	Chrome ferrite, incipient fusion, cleavage carbides or austenite.	103
2 per cent. nickel			
Martensite.	Martensite, coarse.	Martensite, very coarse, possibly incipient delta iron.	92

TABLE 1.—*Constituents of Different Compositions of Chromium-*

Alloy No.	Content			Extra Annealed	Annealed	900° C. Quench
	Cr, Per Cent.	Ni, Per Cent.	C, Per Cent.			
95	14.6	2.1	0.09	Chrome ferrite, martensitic markings.	Martensite, fine.	Martensite.
98	21.4	2.1	0.09	Chrome ferrite, clear, 50 per cent.; martensite 50 per cent.	Same as extra annealed.	Same as annealed.
101	29.3	2.2	0.10	Chrome ferrite, carbides.	Chrome ferrite, carbides in circles marking old austenite lakes.	Same as annealed condition.
105	51.0	2.4	0.18	B constituent, carbides.	B constituent, carbides beaded in grain boundaries.	Chrome ferrite.

IV. Alloys containing

93	11.0	4.0	0.16	Chrome ferrite, martensitic markings.	Chrome ferrite, martensitic markings.	Martensite, fine.
96	14.3	3.5	0.05	Chrome ferrite, martensitic markings.	Chrome ferrite, martensitic markings.	Martensite, fine.
99	20.1	3.0	0.06	Chrome ferrite, clear, 65 per cent.; with martensitic markings, 35 per cent.	Chrome ferrite 65 per cent., austenite 35 per cent.	Same as annealed condition.
102	30.2	3.1	0.26	Chrome ferrite 80 per cent., cleavage austenite 20 per cent., carbides.	Same as extra annealed condition, carbides within austenite.	Same as annealed condition.

V. Alloys containing

2	10.0	4.5	0.12	Chrome ferrite, martensitic markings.	Same as extra annealed.	Same as annealed.
4	15.1	5.3	0.18	Chrome ferrite, martensitic markings.	Same as extra annealed.	Same as annealed condition.
7	22.3	4.8	0.18	Chrome ferrite 65 per cent., austenite 35 per cent., carbides.	Chrome ferrite 65 per cent., austenite 35 per cent.	Austenite 60 per cent., ferrite 40 per cent.
11	24.7	4.9	0.24	Austenite 60 per cent., chrome ferrite 40 per cent.	Same as extra annealed.	Same as annealed except austenite slightly decomposed.
15	28.8	5.1	0.24	Austenite 50 per cent. with carbides, chrome ferrite 50 per cent.	Same as extra annealed.	Same as annealed condition.
16	31.8	5.0	0.32	Chrome ferrite 65 per cent., austenite with carbides 35 per cent.	Same as extra annealed.	Same as annealed condition.
106	35.6	5.0	0.14	Chrome ferrite 90 per cent., cleavage austenite 10 per cent.	Same as extra annealed.	Chrome ferrite, carbides.
26	43.0	5.5	0.30	B constituent, carbides.	Same as extra annealed.	Same as annealed condition.

nickel-iron Alloys after Different Treatments.—(Continued)

1050° C. Quench	1200° C. Quench	1300° C. Quench	Alloy No.
Same as 900° C. quench.	Martensite, coarse.	Martensite, very coarse.	95
Chrome ferrite 40 per cent., martensite 60 per cent.	Chrome ferrite 65 per cent. with recrystallization, martensite coarse 35 per cent.	Chrome ferrite, very coarse; cleavage carbide or austenite.	98
Chrome ferrite 97 per cent., cleavage austenite 3 per cent.	Chrome ferrite 40 per cent., austenite 60 per cent.	Chrome ferrite, coarse; carbide reprecipitation. Thin line of austenite in grain boundaries.	101
Chrome ferrite.	Chrome ferrite, recrystallization, cleavage austenite.	Chrome ferrite, coarse; recrystallization and reprecipitation of carbides, incipient fusion.	105
about 4 per cent. nickel			
Martensite, fine.	Martensite, fine.	Martensite, coarse.	93
Martensite, fine.	Martensite, coarse.	Martensite, very coarse.	96
Chrome ferrite, coarse, 60 per cent.; austenite 40 per cent.	Same as 1050° quench.	Chrome ferrite, very coarse; cleavage austenite.	99
Chrome ferrite, recrystallized, 80 per cent.; austenite 20 per cent.	Chrome ferrite, coarsened, 80 per cent.; austenite 20 per cent.	Chrome ferrite 80 per cent., austenite 20 per cent.	102
about 5 per cent. nickel			
Martensite.	Martensite.	Martensite, coarse.	2
Martensite.	Martensite, coarse.	Martensite, very coarse.	4
Austenite 50 per cent., chrome ferrite 50 per cent.	Austenite, coarse, 50 per cent.; chrome ferrite, coarse 50 per cent.; phases thoroughly divorced.	Austenite 35 per cent., chrome ferrite 65 per cent. very coarse, recrystallization and carbide precipitation, ferrite cleavages.	7
Austenite 50 per cent., chrome ferrite 50 per cent.	Austenite 50 per cent., chrome ferrite 50 per cent.	Austenite 50 per cent., chrome ferrite 50 per cent., coarsened austenite continuous.	11
Chrome ferrite recrystallized 65 per cent., austenite 35 per cent.	Same as 1050° C. quench.	Chrome ferrite, coarse, 80 per cent.; austenite in cleavages 20 per cent.; carbides reprecipitated in ferrite.	15
Chrome ferrite recrystallized 70 per cent., austenite 30 per cent.	Chrome ferrite 90 per cent., austenite 10 per cent.; carbides reprecipitated in ferrite.	Chrome ferrite, very coarse, 98 per cent.; cleavage austenite 2 per cent., incipient fusion.	16
Chrome ferrite, reprecipitated carbides.	Same as 1050° C. quench.	Chrome ferrite, very coarse; carbides reprecipitated in cleavages.	106
Chrome ferrite, carbides.	Chrome ferrite, recrystallization, carbides.	Chrome ferrite, very coarse; incipient fusion, cleavage carbides.	26

TABLE 1.—*Constituents of Different Compositions of Chromium-*

Alloy No.	Content			Extra Annealed	Annealed	900° C. Quench
	Cr, Per Cent.	Ni, Per Cent.	C, Per Cent.			
38	49.1	5.2	0.42	<i>B</i> constituent, carbides.	<i>B</i> constituent, carbides.	Same as annealed condition.
86	44.6	5.0	0.15	<i>B</i> constituent, carbides	<i>B</i> constituent, carbides.	Same as annealed condition.
45	53.6	4.7	0.33	<i>B</i> constituent massive 70 per cent., sorbitic with ferrite 30 per cent.	Chrome ferrite, carbides.	Same as annealed condition.
87	55.5	4.9	0.19	Chrome ferrite carbides or austenite.	Same as extra annealed, finer carbides.	Same as annealed condition.

VI. Alloys containing

5	10.2	9.5	0.24	Chrome ferrite, martensitic markings.	Austenite, three-quarters transformed to martensite (martensite 75 per cent., austenite 25 per cent.).	Austenite, half transformed to martensite (martensite 50 per cent.; austenite 50 per cent.).
8	15.7	9.9	0.12	Austenite, slightly decomposed, carbide precipitation.	Austenite, medium grained	Same as annealed condition.
12	19.1	9.9	0.20	Austenite with excess carbide precipitated in parallel lines.	Austenite, medium grained.	Same as annealed condition.
17	24.6	9.7	0.20	Austenite, carbide precipitation in cleavages.	Austenite, medium grained.	Same as annealed condition.
22	28.6	10.3	0.22	Austenite 80 per cent., <i>B</i> constituent 20 per cent.	Same as extra annealed.	Austenite 70 per cent., fine; <i>B</i> constituent 30 per cent.
27	34.5	11.6	0.16	<i>B</i> constituent 55 per cent. (some troostitic), austenite 45 per cent.	Same as extra annealed.	Same as annealed condition.
32	41.4	9.3	0.32	<i>B</i> constituent 70 per cent., austenite 30 per cent.	Same as extra annealed.	Same as annealed condition.
39	46.1	10.0	0.40	<i>B</i> constituent 60 per cent., austenite 40 per cent., carbides.	<i>B</i> constituent sorbitic 60 per cent., austenite 40 per cent.	Same as annealed condition.
46	50.0	10.1	0.36	<i>B</i> constituent 80 per cent., cleavage austenite 20 per cent.	<i>B</i> constituent sorbitic 80 per cent., austenite 20 per cent., spheroidized carbides.	Chrome ferrite 80 per cent., austenite 20 per cent.
88*	50.8	10.3	0.25			

VII. Alloys containing

9	9.8	14.7	0.18	Austenite, 5 per cent. transformed to needlelike ferrite.	Austenite, fine	Same as annealed condition.
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* Essentially as alloy No. 46.

nickel-iron Alloys after Different Treatments.—(Continued)

1050° C. Quench	1200° C. Quench	1300° C. Quench	Alloy No.
Chrome ferrite, carbides.	Chrome ferrite, recrystallization, reprecipitated carbides in cleavages.	Chrome ferrite, very coarse; incipient fusion.	38
Chrome ferrite, carbides.	Chrome ferrite, coarse, recrystallization.	Chrome ferrite, incipient fusion, carbides in cleavages and grain boundaries.	86
Same as 900° C. quench.	Chrome ferrite coarse, recrystallization, reprecipitation of carbides and coalescence.	Same as 1200° C. quench.	45
Same as 900° C. quench.	Chrome ferrite, coarse; carbide reprecipitation and coalescence.	Chrome ferrite, cleavage carbides or austenite, incipient fusion.	87

about 10 per cent. nickel

Same as 900° C. quench.	Austenite, coarse three-quarters transformed to martensite (martensite 75 per cent., austenite 25 per cent.).	Martensite, coarse.	5
Same as annealed condition.	Austenite, coarse.	Austenite, very coarse, slightly decomposed to ferrite needles.	8
Same as annealed condition.	Same as annealed condition.	Austenite coarse	12
Same as annealed condition.	Austenite 95 per cent., chrome ferrite 5 per cent.	Austenite 80 per cent., chrome ferrite 20 per cent.	17
Austenite, fine, 50 per cent.; chrome ferrite 50 per cent.	Austenite, fine, 50 per cent.; chrome ferrite 50 per cent.; austenite contains fine precipitate.	Austenite, coarse, 50 per cent.; chrome ferrite, coarse, 50 per cent.	22
Austenite, fine, 45 per cent.; chrome ferrite 55 per cent.	Chrome ferrite 70 per cent., austenite 30 per cent.	Chrome ferrite reprecipitated and coalesced carbides.	27
Chrome ferrite 70 per cent., austenite 30 per cent.	Chrome ferrite 90 per cent., cleavage austenite 10 per cent., carbide reprecipitation in ferrite.	Chrome ferrite very coarse, incipient fusion.	32
Chrome ferrite 90 per cent., austenite 10 per cent., carbides.	Chrome ferrite recrystallized, carbides.	Chrome ferrite very coarse, incipient fusion, carbides coalesced.	39
Chrome ferrite recrystallized.	Chrome ferrite coarse, carbide reprecipitation.	Chrome ferrite very coarse, incipient fusion, carbides coalesced.	46
			88

about 15 per cent. nickel

Same as annealed condition.	Austenite, coarse.	Austenite, very coarse, one-quarter transformed to chrome ferrite (austenite 75 per cent., chrome ferrite 25 per cent.).	9
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TABLE 1.—*Constituents of Different Compositions of Chromium-*

Alloy No.	Content			Extra Annealed	Annealed	900° C. Quench
	Cr, Per Cent.	Ni, Per Cent.	C, Per Cent.			
13	14.1	13.8	0.12	Austenite, fine.	Same as extra annealed.	Austenite, medium grained.
18	19.8	14.9	0.24	Austenite, slight precipitation carbides in plates.	Austenite, fine.	Austenite, medium grain.
23	24.6	15.1	0.44	B constituent 30 per cent., austenite 70 per cent.	Same as extra annealed.	Austenite.
85	25.0	14.9	0.14	Austenite, carbides precipitated slightly.	Austenite, medium grain	Same as annealed condition.
28	29.6	13.7	0.22	B constituent 20 per cent., austenite 80 per cent.	Same as extra annealed.	Same as annealed condition.
33	34.3	14.6	0.20	B constituent 35 per cent., austenite 65 per cent.	B constituent-sorbite 35 per cent., austenite 65 per cent.	Same as annealed condition.
40	43.7	13.7	0.41	B constituent 35 per cent., austenite 65 per cent.	B constituent sorbitic 35 per cent., austenite 65 per cent.	Same as annealed condition.

VIII. Alloys containing

19	14.3	20.3	0.17	Austenite, slight precipitation carbides, parallel plates.	Austenite, fine.	Same as annealed condition.
24	21.9	19.9	0.18	Austenite, medium grain.	Same as extra annealed.	Same as annealed condition.
29	24.9	20.5	0.18	Austenite, medium grain.	Same as extra annealed.	Same as annealed condition.
34	32.7	21.5	0.24	Austenite, carbide precipitate, parallel plates.	Austenite, medium grain.	Same as annealed condition.
41	36.6	20.5	0.48	B constituent sorbitic 20 per cent., austenite 80 per cent.	Same as extra annealed.	Same as annealed condition.

IX. Alloys containing

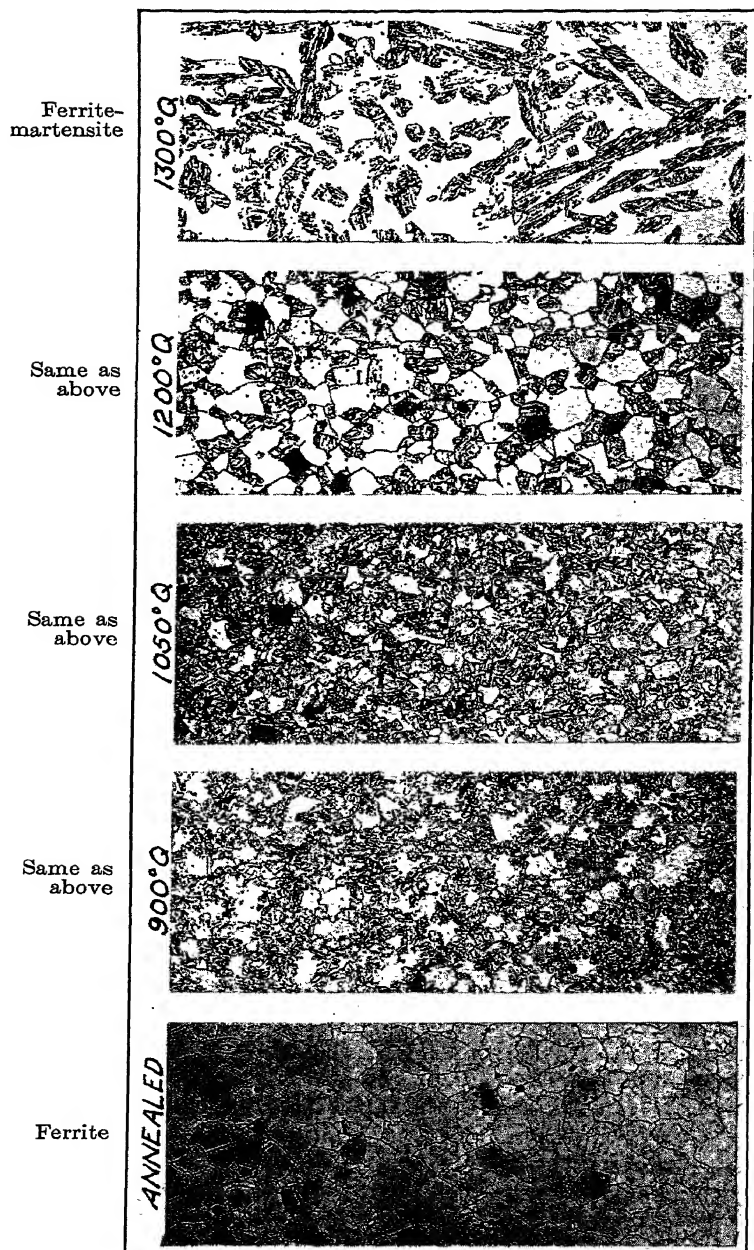
30	21.0	24.5	0.12	Austenite, medium grain.	Same as extra annealed.	Same as annealed condition.
35	24.4	24.2	0.29	Austenite carbides, precipitated parallel plates.	Austenite, medium grain.	Same as annealed condition.
42	28.4	26.0	0.18	Austenite fine.	Same as extra annealed.	Same as annealed condition.
47	38.9	22.5	0.18	B constituent 25 per cent. (continuous network), austenite 75 per cent.	Same as extra annealed.	Same as annealed condition.

X. Alloys containing

36	18.6	30.5	0.14	Austenite, medium grain, slight excess carbide precipitation.	Austenite, medium grain.	Same as annealed condition.
43	25.4	30.6	0.22	Austenite, spheroidized carbides.	Same as extra annealed.	Austenite, medium grain.

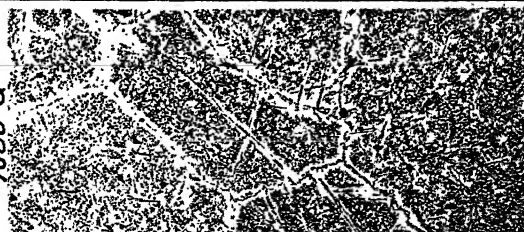
nickel-iron Alloys after Different Treatments.—(Continued)

1050° C. Quench	1200° C. Quench	1300° C. Quench	Alloy No.
Same as 900° C. quench.	Same as 900° C. quench.	Austenite, coarse.	13
Same as 900° C. quench.	Same as 900° C. quench.	Austenite, coarse.	18
Austenite.	Austenite, coarse.	Austenite, very coarse, incipient fusion.	23
Same as annealed condition.	Austenite, coarse.	Austenite, coarse.	85
Austenite fine 80 per cent., chrome ferrite 20 per cent.	Austenite 80 per cent., chrome ferrite 20 per cent.	Austenite, coarse, 80 per cent.; chrome ferrite 20 per cent.	28
Austenite 65 per cent., chrome ferrite 35 per cent.	Austenite, coarse 65 per cent., chrome ferrite 35 per cent.	Austenite, coarse, 60 per cent.; chrome ferrite 40 per cent.	35
Austenite 65 per cent., chrome ferrite 35 per cent.	Same as 1050° C. quench.	Chrome ferrite 80 per cent., austenite 20 per cent.	40
about 20 per cent. nickel			
Same as annealed condition.	Austenite, medium grain.	Austenite, coarse.	19
Same as annealed condition.	Same as annealed condition.	Austenite, coarse.	24
Same as annealed condition.	Austenite, coarse.	Austenite, very coarse.	29
Same as annealed condition.	Austenite, coarse.	Austenite, very coarse.	34
Austenite fine 80 per cent., chrome ferrite 20 per cent.	Same as 1050° C. quench.	Austenite coarse 80 per cent., chrome ferrite 20 per cent.	41
about 25 per cent. nickel			
Same as annealed condition.	Austenite, coarse.	Austenite, very coarse.	30
Same as annealed condition.	Austenite, coarse.	Austenite, very coarse.	35
Austenite, medium grain.	Same as 1050° C. quench.	Austenite, coarse.	42
Austenite 50 per cent., chrome ferrite 50 per cent.	Same as 1050° C. quench.	Chrome ferrite 60 per cent., austenite 40 per cent.	47
about 30 per cent. nickel			
Same as annealed condition.	Austenite, coarse.	Austenite, very coarse.	36
Same as 900° C. quench.	Same as 900° C. quench.	Austenite, very coarse.	43

FIG. 7.—FERRITE-MARTENSITE ALLOYS. $\times 75$.

Ferrite and
austenite

1300°Q

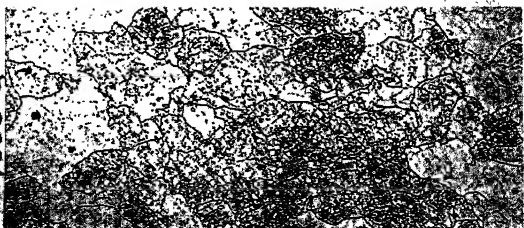
Same as
above

1200°Q



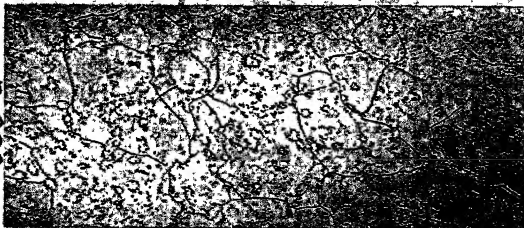
Ferrite

1050°Q



Ferrite

900°Q



Ferrite

ANNEALED

FIG. 8 — FERRITE-AUSTENITE ALLOYS $\times 75$

Martensitic Alloys

Surprisingly, a number of alloys have just such a balance of tendencies between transformation and sluggishness that under almost all conditions they exist in a martensitic form. With annealing treatment approaching

COMPOSITION OF MARTENSITIC ALLOYS							
ALLOY No.	CHROMIUM, PER CENT.	NICKEL, PER CENT.	CARBON, PER CENT.	ALLOY No.	CHROMIUM, PER CENT.	NICKEL, PER CENT.	CARBON PER CENT.
91	10.3	1.1	0.08	96	14.3	3.5	0.05
92	11.0	2.2	0.07	2	10.0	4.5	0.12
95	14.6	2.1	0.09	4	15.1	5.3	0.18
93	11.0	4.0	0.16				

infinite time, they would presumably become polyhedral chrome-ferrite, but even though they soften by annealing, characteristic acicular markings persist. The older charts, which indicated certain alloys as martensitic, truly had a sound basis for their construction even though their precise regional locations may have been in error. The alloys analyzed that have relatively low carbon do not become steel-hard but are nevertheless enormously hardened by the quench. Fig. 6, alloy 93, is very characteristic of the group.

Ferrite-martensite Group

COMPOSITION OF FERRITE-MARTENSITE			
ALLOY No.	CHROMIUM, PER CENT.	NICKEL, PER CENT.	CARBON, PER CENT.
1	14.3	Trace	0.18
3	20.2	Trace	0.12
94	5.9	1.2	0.05
97	19.5	1.1	0.08
98	21.4	2.1	0.09

The basis for the development of this group is the chromium effect to render alpha iron stable and non-transforming when present in about 13 per cent. excess. This effect is counteracted by carbon and nickel which extend the "austenite loop" into richer chromium alloys. The result then is the formation of two-phase alloys, one portion of which is perpetually ferrite with the other undergoing the transformation to austenite. When the alloy content of this austenite phase is not great, it is rendered martensitic during the cooling. Fig. 7, alloy 1, is characteristic of the group. The rustless-iron commercial alloys are in general of this sort unless the carbon is relatively high and the chromium low.

Austenite-ferrite Alloys

Except for the increased stability of the austenite phase in this group, developed by higher alloy content, it is substantially like the preceding group. The outstanding behavior of these alloys is the increase of austenite in low-quenched specimens over that found in the annealed or

COMPOSITION OF AUSTENITE-FERRITE ALLOYS

ALLOY No.	CHROMIUM, PER CENT.	NICKEL, PER CENT.	CARBON, PER CENT.	ALLOY No.	CHROMIUM, PER CENT.	NICKEL, PER CENT.	CARBON, PER CENT.
100	29.8	1.2	0.09	11	24.7	4.9	0.24
101	29.3	2.2	0.10	15	28.8	5.1	0.24
99	20.1	3.0	0.06	16	31.8	5.0	0.32
102	30.2	3.1	0.26	17	24.6	9.7	0.20
7	22.3	4.8	0.18				

high-quenched condition. Here again is reflected the "austenite loop" tendency. At higher or lower temperatures, alpha iron is dominating the structure, while the temperature zone of greatest austenite extension makes itself felt in intermediate temperatures. Fig. 8 is fairly characteristic of such alloys, although in general some austenite may be either at the lowest anneal or the highest quench. In the ordinary chrome irons, which contain about 22 per cent. chromium and about 0.35 per cent. carbon, the same effects are to be found. Nickel serves to retain a little austenite in all conditions.

The recession of the large austenite pools developed at intermediate temperature is responsible for a curious appearance found in so many alloys of this study, particularly in this and the group just preceding. The portion of metal that has had the first allotropic change to austenite followed by the reversal is lined with a set of fine imperfect grain boundaries designated as recrystallization traces. Wherever either diffusion of a constituent or an allotropic change takes place these secondary grain systems seem to be traced permanently. Furthermore, a deposit of fine carbide particles formerly in solution in the austenite is always observed. They are plainly to be seen in Fig. 8. In general it is almost impossible to say with assurance whether carbide particles or fragments of austenite are left after the major portion of a region has undergone change from austenite. At high temperature, in every case the shrinking austenite forms a set of very fine plates in Widmanstätten distribution, Fig. 9. The various grains etch differently depending upon the relation of these cleavage planes to the plane of polished section.

The Wholly Ferritic Alloys

This grouping is made with some reservation although strictly in accord with the experimental results. This much may be said however; all the alloys tested are dominantly ferritic at all temperatures considered. There are two possible points of vagueness here. In the first

COMPOSITION OF ALLOYS WHOLLY FERRITIC

ALLOY No.	CHROMIUM, PER CENT.	NICKEL, PER CENT.	CARBON, PER CENT.	ALLOY No.	CHROMIUM, PER CENT.	NICKEL, PER CENT.	CARBON, PER CENT.
6	26.8	0.3	0.24	44	58.0	0.2	0.36
10	28.9	0.4	0.18	20	41.3	1.0	0.36
14	36.8	0.6	0.17	106	35.6	5.0	0.14
31	51.2	0.3	0.38	87	53.6	4.7	0.33
37	56.6	0.3	0.42				

place, in some of the alloys the carbides seemed prone to enlarge in size so as to form an appreciable austenite phase, which may amount to 10 per cent. of the total volume at a maximum. The other consideration is that some of these compositions might undergo the transformation to the brittle constituent, if enough time were allowed. In particular, alloy 20 (Cr, 41.3 per cent., Ni, 1.0 per cent., C, 0.36 per cent.), appeared to have a few nuclei of transformed material. However, the long anneal did not bring about the change. The continuous uniform increase in hard-

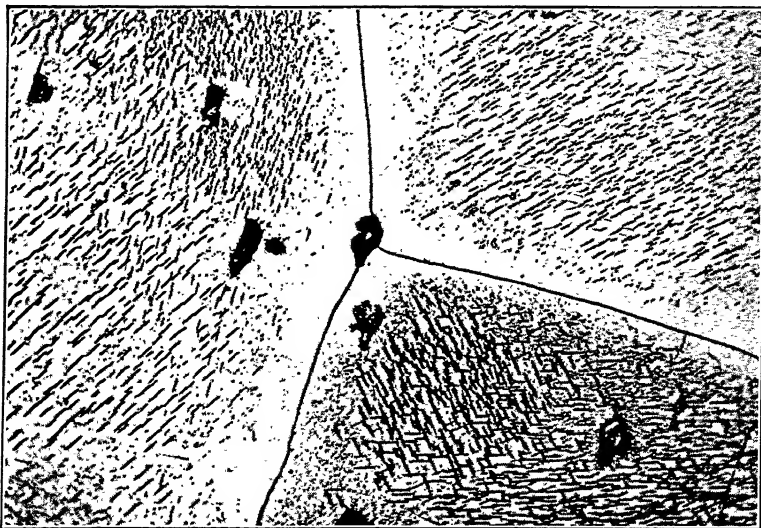


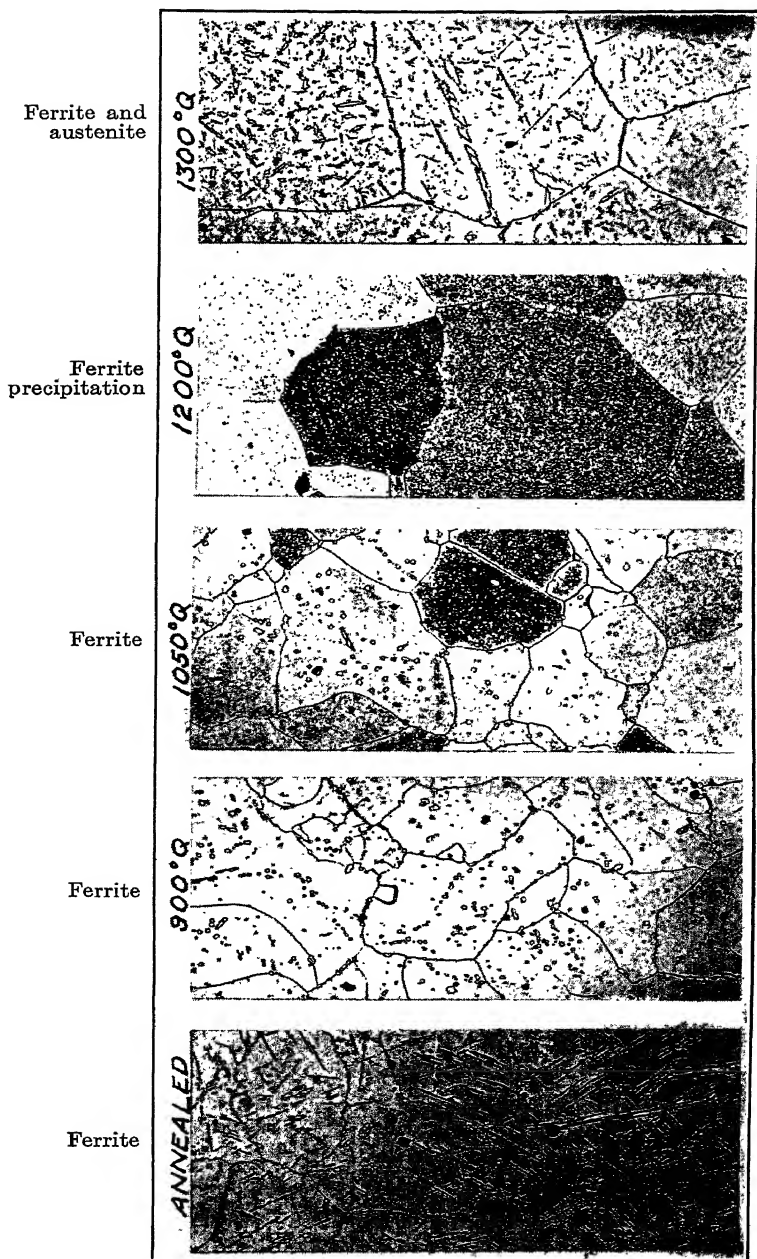
FIG. 9.—CLEAVAGE PLANE PRECIPITATION OF AUSTENITE FROM FERRITE AT VERY HIGH TEMPERATURE. $\times 1000$.

When coarser, it resembles a Widmanstätten structure.

ness of these alloys, with increase in chromium content, from 19 to 41 Rockwell "C," forms a striking example of solid-solution hardness. The presence (at about 50 per cent. chromium) of 5 per cent. nickel raises the hardness to 52 Rockwell "C" without introducing a new phase. Figs. 10 and 11 show the structures found in this group.

The Brittle Constituent Alloys

This group comprises those compositions which, with sufficiently long annealing, undergo complete atomic rearrangement from ferrite to the complex crystal form possessing great hardness. Usually an attempt to estimate the hardness with the Rockwell tester with diamond point results in shattering and splitting out fragments of the metal.

FIG. 10.—FERRITE ALLOYS. $\times 75$.

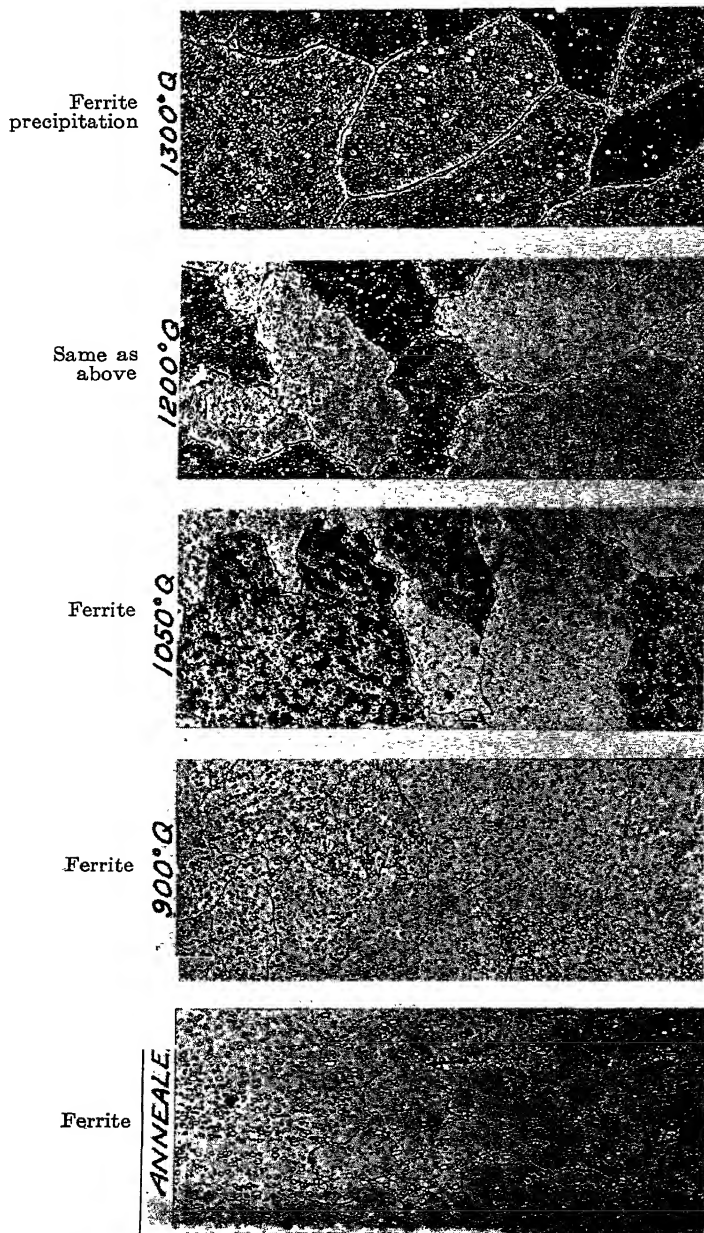
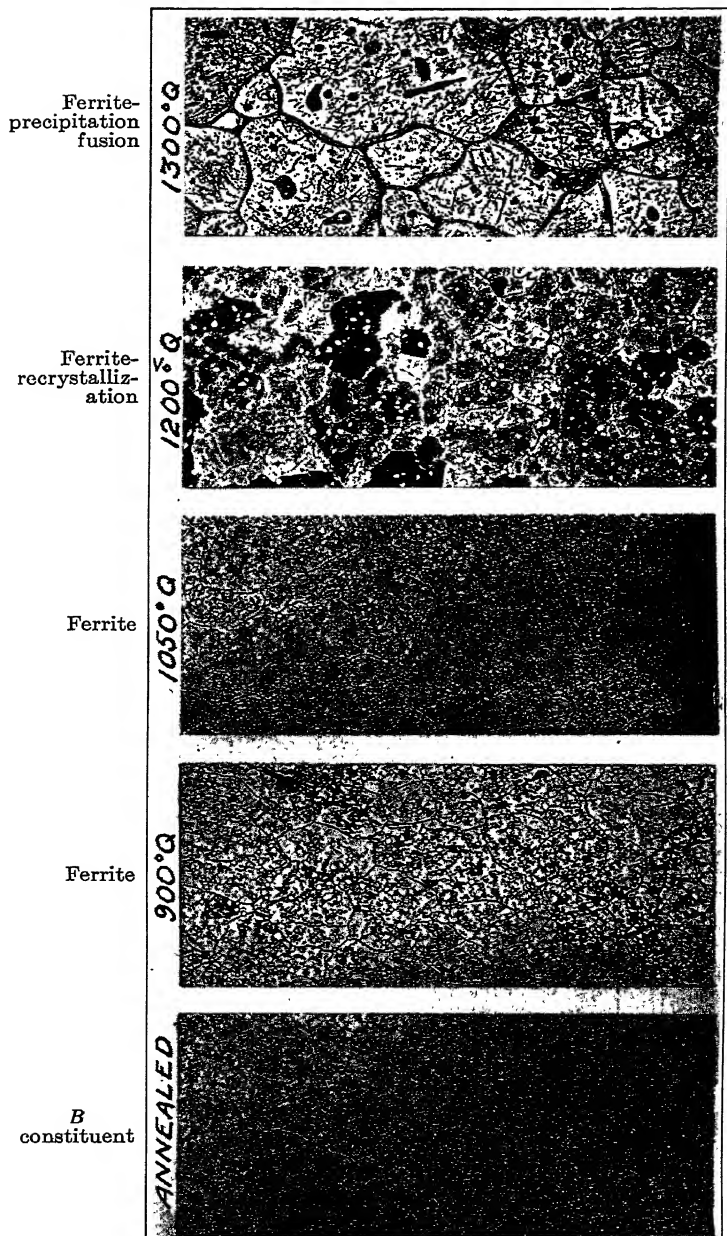


FIG. 11.—FERRITE ALLOYS IN WHICH THE CARBIDES ARE DIFFUSED AND REPRECIPITATED. $\times 75$.



3. 12.—THE B-CONSTITUENT ALLOYS SHOWING TRANSFORMATION TO FERRITE AT ELEVATED TEMPERATURES. $\times 75$.

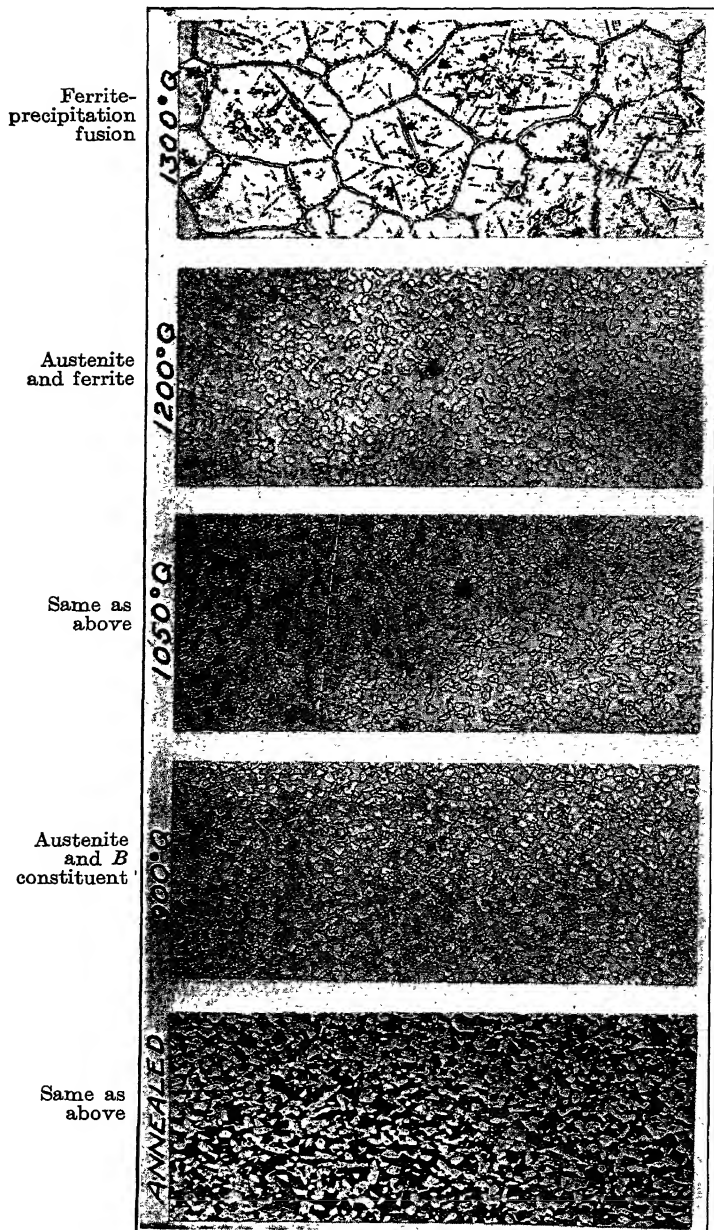


FIG 13.—THE B-CONSTITUENT AUSTENITE ALLOYS IN WHICH FERRITE GREATLY PRE-
DOMINATES AT ELEVATED TEMPERATURE. $\times 75$.

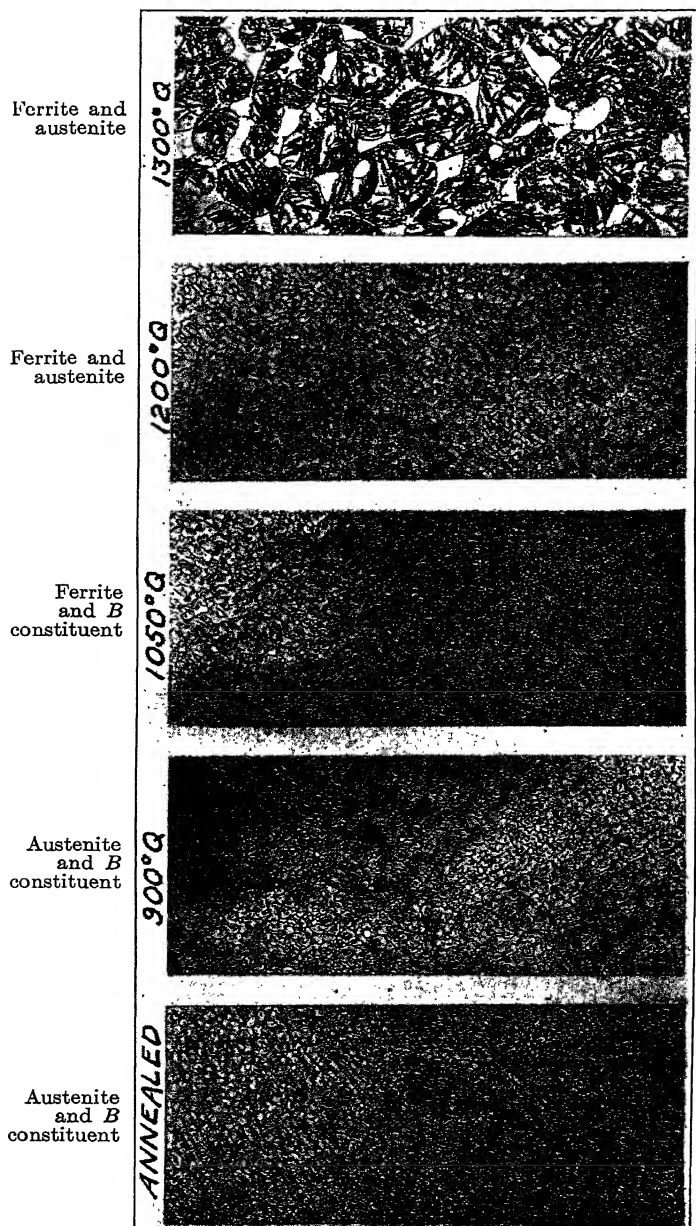


FIG 14.—THE B-CONSTITUENT AUSTENITE ALLOYS IN WHICH AUSTENITE IS VOLUMINOUS. $\times 75$.

COMPOSITION OF ALLOYS WITH THE BRITTLE CONSTITUENT							
ALLOY No.	CHROMIUM, PER CENT.	NICKEL, PER CENT.	CARBON, PER CENT.	ALLOY No.	CHROMIUM, PER CENT.	NICKEL, PER CENT.	CARBON, PER CENT.
25	47.3	0.3	0.44	38	49.1	5.2	0.42
103	52.1	1.0	0.31	86	44.6	5.0	0.15
105	51.0	2.4	0.18	45	53.6	4.7	0.33
26	43.0	5.5	0.30	88	50.3	10.3	0.25

Such a reading is meaningless except to show friability. However, in a few instances very proper readings of 68.5 “*C*” were obtained. As forged, the alloys are ferritic, and even a moderately slow cooling in air from the forging temperature preserves this magnetic condition. A few hours at 800° C. for this group, however, causes sufficient atomic mobility to cause complete embrittling by transformation. It is difficult to discriminate between the compositions that will develop the *B* constituent and that will not, for each large increment in time added one or two more alloys to the list. Alloy 31 probably would eventually revert to the *B* constituent. Fig. 12 shows the microstructure.

Austenite—B-constituent Alloys

In general, the *B* constituent is never associated as a phase along with ferrite; once the recrystallization has started it proceeds across the homogeneous grains to completion in one of two modes. But with certain compositions there is an equilibrium established between ferrite

COMPOSITION OF ALLOYS WITH AUSTENITE— <i>B</i> CONSTITUENT				
ALLOY No.	CHROMIUM, PER CENT.	NICKEL, PER CENT.	CARBON, PER CENT.	<i>B</i> CONSTITUENT, PER CENT.
22	28.6	10.3	0.22	20
27	34.5	11.6	0.16	55
32	41.4	9.3	0.32	70
39	46.1	10.0	0.40	60
46	50.0	10.1	0.36	80
23	24.6	15.1	0.44	30
28	29.6	13.7	0.22	20
33	34.3	14.6	0.20	35
40	43.7	13.7	0.41	35
41	36.6	20.5	0.48	20
47	38.9	22.5	0.18	25

and austenite, the ferrite phase of which is of a composition to permit of the allotropic change to the *B* constituent. This constituent then transforms to ferrite upon heating above the critical temperature as when it is a single phase alone. These compositions, therefore, are austenite-*B*-constituent materials at low temperatures and austenite-ferrite mixtures at elevated temperatures and, as has been pointed out in connection with the other austenite-ferrite alloys, the ferrite phase usually becomes dominant near the melting point. Figs. 13 and 14 are characteristic of this group.

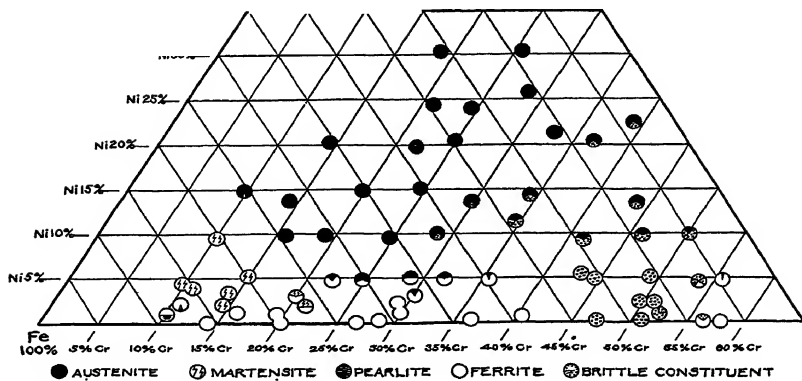


FIG. 15.—STRUCTURES FOUND IN THE ALLOYS AFTER LONG ANNEALING.

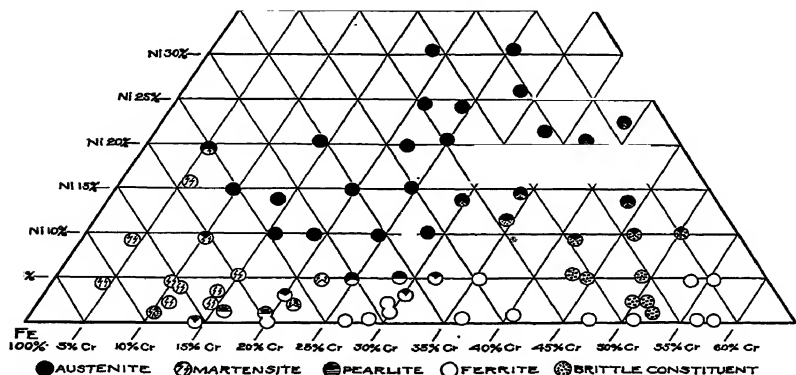


FIG. 16.—STRUCTURES FOUND IN THE ALLOYS AFTER MODERATE ANNEALING.

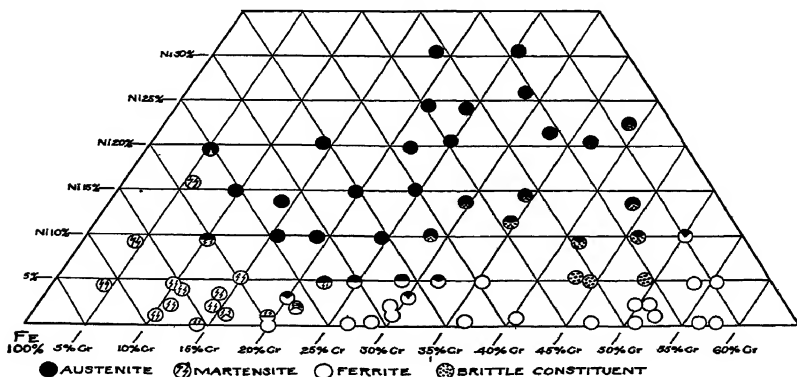


FIG. 17.—STRUCTURES FOUND IN THE ALLOYS AFTER 900° C. QUENCH.

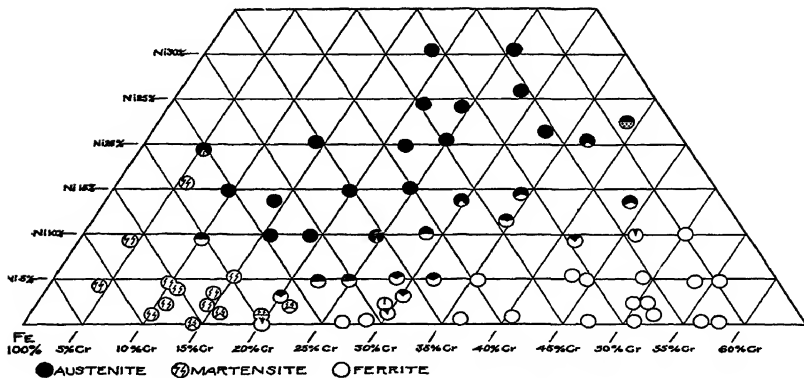


FIG. 18.—STRUCTURES FOUND IN THE ALLOYS AFTER 1050° C. QUENCH.

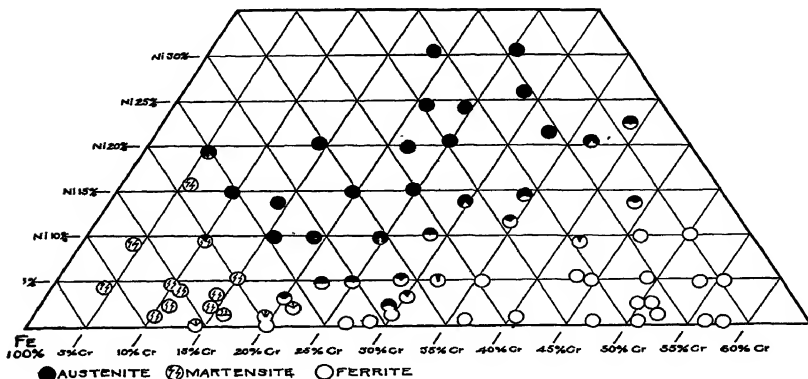


FIG. 19.—STRUCTURES FOUND IN THE ALLOYS AFTER 1200° C. QUENCH.

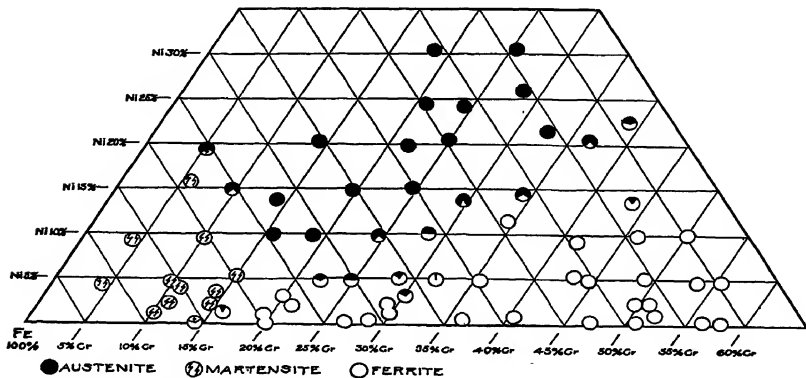
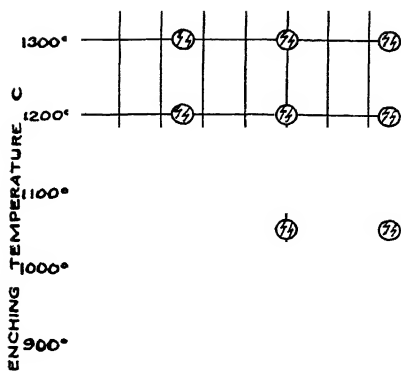
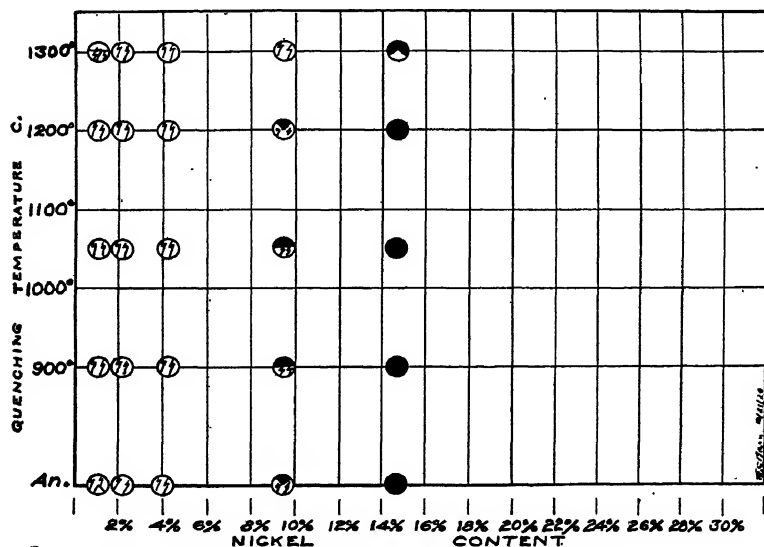


FIG. 20.—STRUCTURES FOUND IN THE ALLOYS AFTER 1300° C. QUENCH.



○ CHROME-FERRITE (1) MARTENSITE (2) AUSTENITE

G. 21.—STRUCTURES FOUND IN ALLOYS CONTAINING APPROXIMATELY 5 PER CENT. CHROMIUM.



○ CHROME-FERRITE (1) MARTENSITE (2) AUSTENITE

G. 22.—STRUCTURES FOUND IN ALLOYS CONTAINING APPROXIMATELY 10 PER CENT. CHROMIUM.

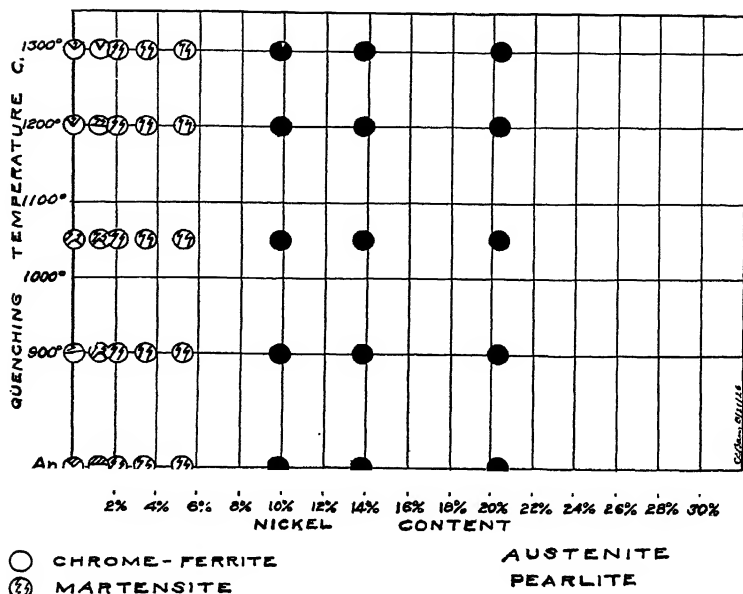


FIG. 23.—STRUCTURES FOUND IN ALLOYS CONTAINING APPROXIMATELY 15 PER CENT CHROMIUM.

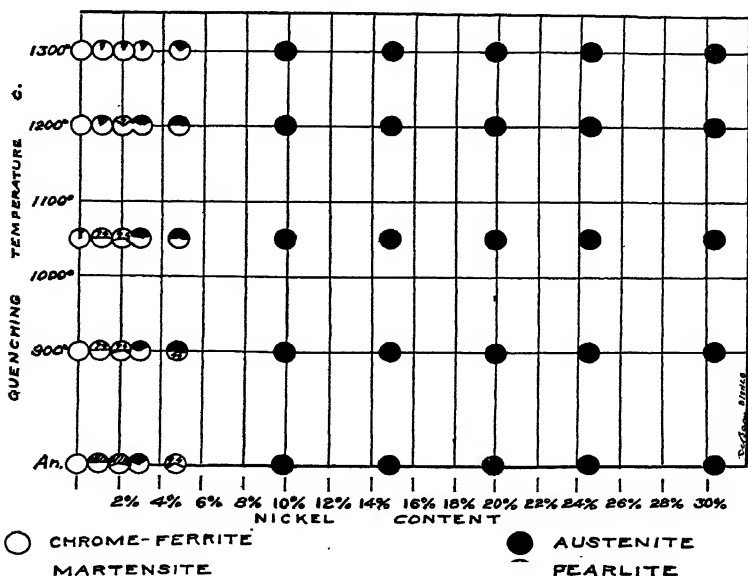


FIG. 24.—STRUCTURES FOUND IN ALLOYS CONTAINING APPROXIMATELY 20 PER CENT CHROMIUM.

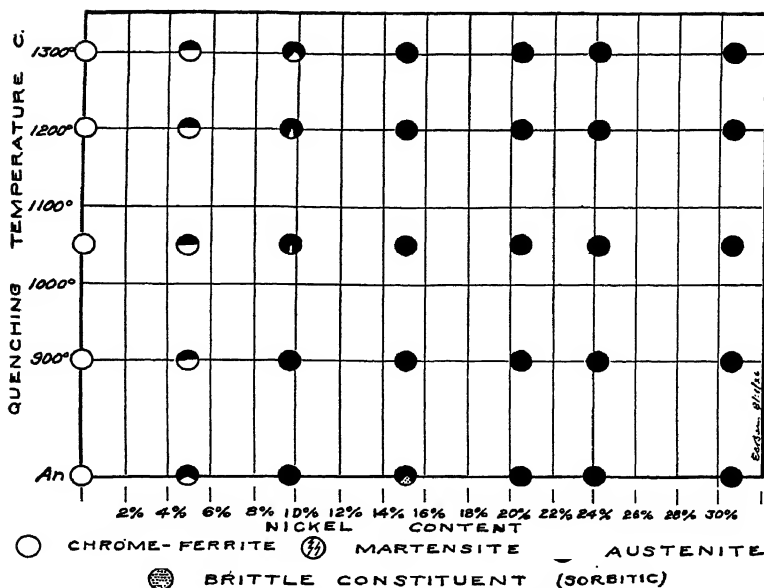


FIG. 25.—STRUCTURES FOUND IN ALLOYS CONTAINING APPROXIMATELY 25 PER CENT CHROMIUM.

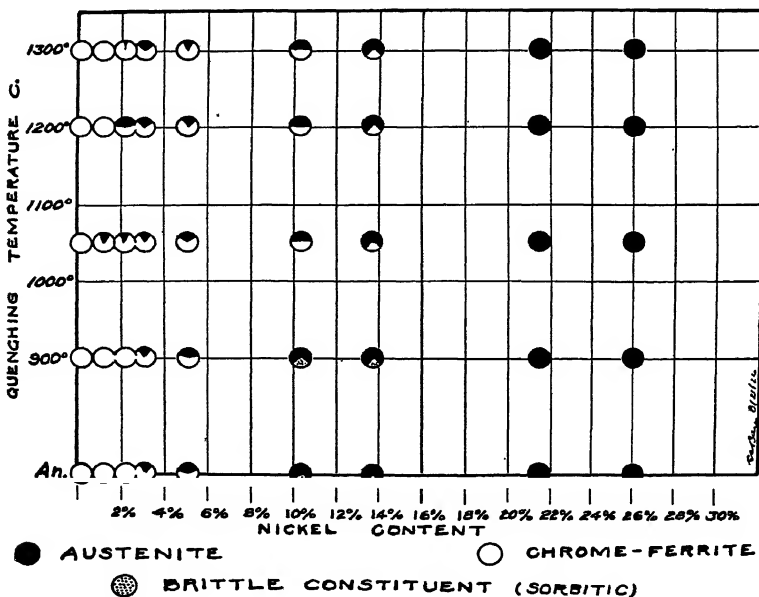


FIG. 26.—STRUCTURES FOUND IN ALLOYS CONTAINING APPROXIMATELY 30 PER CENT CHROMIUM.

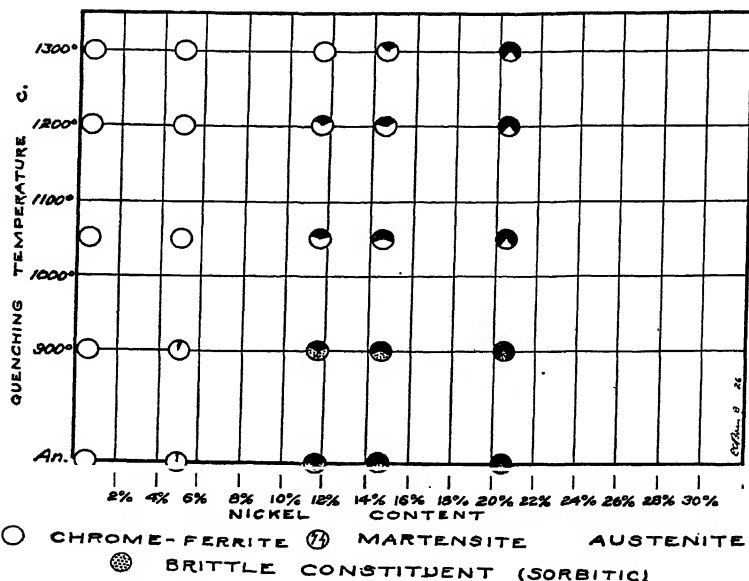


FIG. 27.—STRUCTURES FOUND IN ALLOYS CONTAINING APPROXIMATELY 35 PER CENT. CHROMIUM.

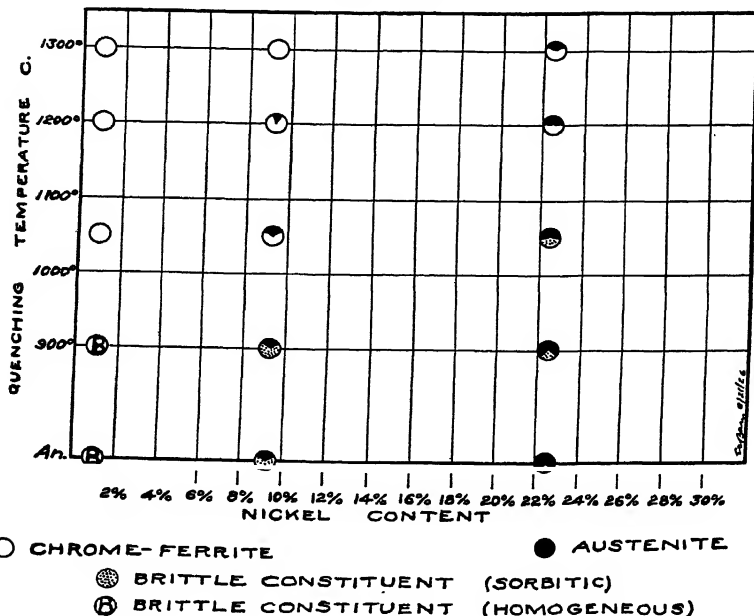


FIG. 28.—STRUCTURES FOUND IN ALLOYS CONTAINING APPROXIMATELY 40 PER CENT. CHROMIUM.

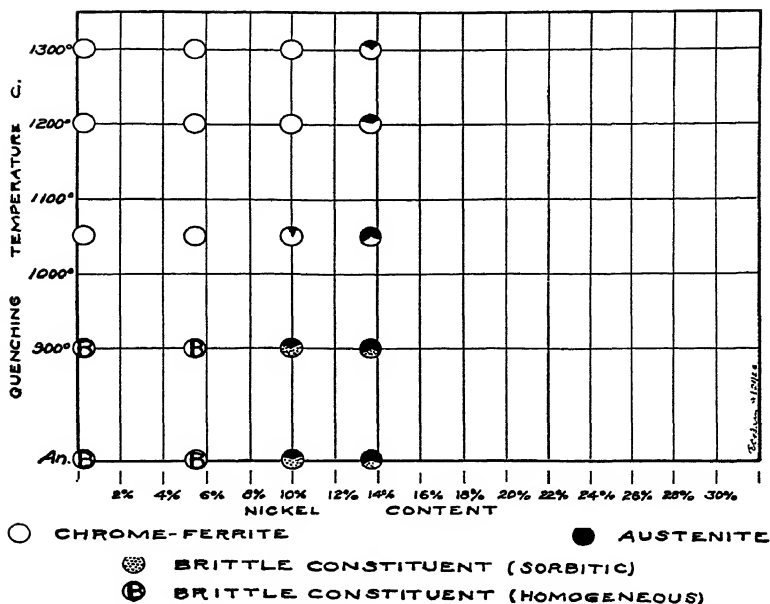


FIG. 29.—STRUCTURES FOUND IN ALLOYS CONTAINING APPROXIMATELY 45 PER CENT. CHROMIUM.

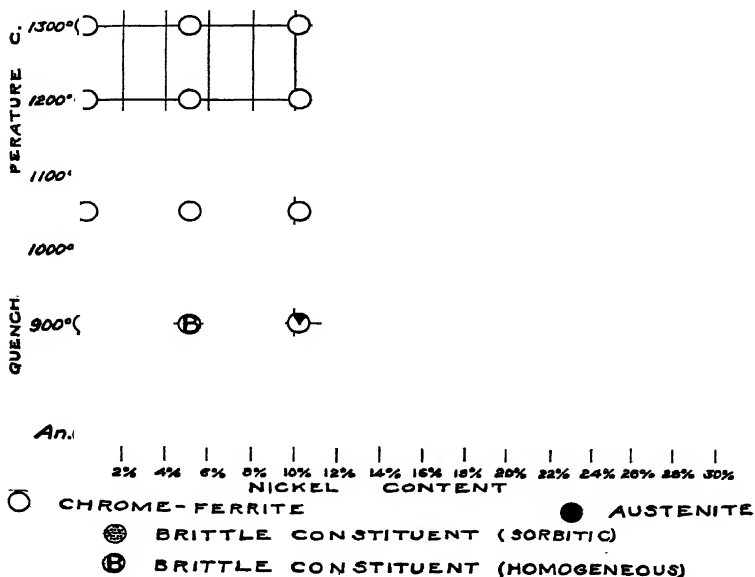


FIG. 30.—STRUCTURES FOUND IN ALLOYS CONTAINING APPROXIMATELY 50 PER CENT. CHROMIUM.

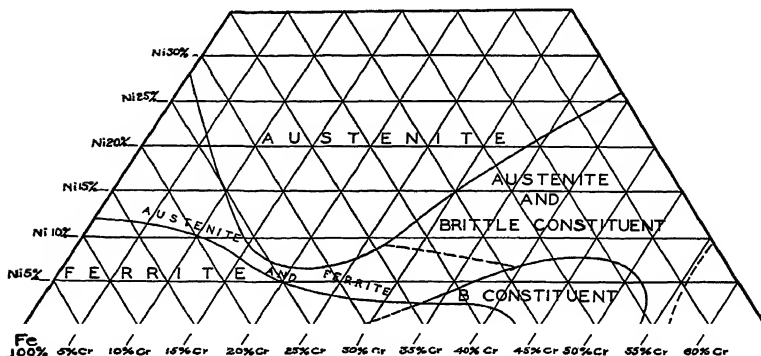


FIG. 31.—PHASE LOCATIONS AFTER LONG ANNEALING, APPROXIMATING EQUILIBRIUM CONDITIONS FOR LOWEST TEMPERATURES.

Note ambiguous behavior in low-nickel alloys from 30 to 43 per cent. chromium.

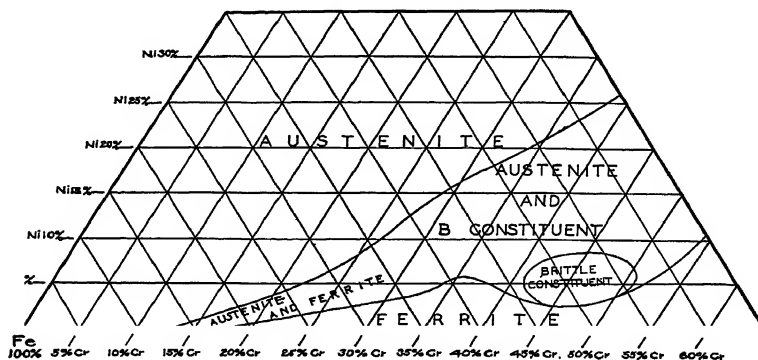


FIG. 32.—PHASE LOCATIONS AT 900° C.

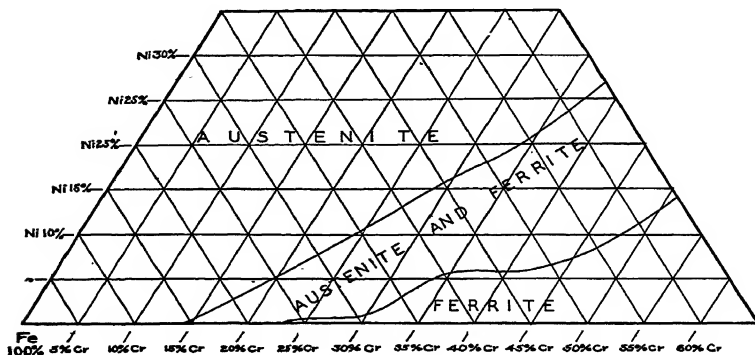


FIG. 33.—PHASE LOCATIONS AT 1050° C.

STRUCTURE CHARTS

The charts on the foregoing pages present the information briefly tabulated above. In Figs. 15 to 20, the basis for each chart is a single heat-treatment applied to all the compositions studied. They are

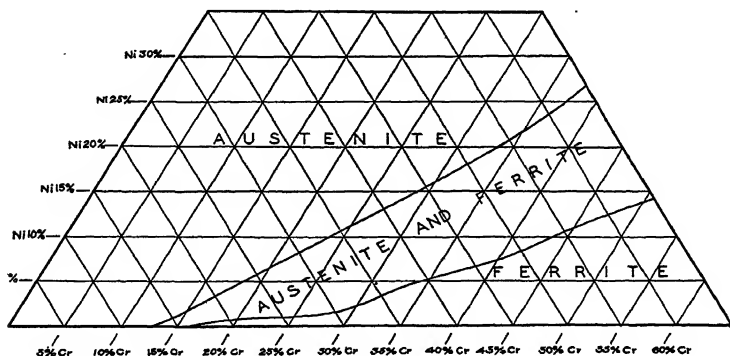


FIG. 34.—PHASE LOCATIONS AT 1200° C.

plotted on the triangular coordinates and symbols denote the structures observed. In Figs. 21 to 30, inclusive, the information is presented in a manner more easily available for practical use. Here the basis is the effect of nickel (abscissa) and heat-treatment (ordinate) upon alloys

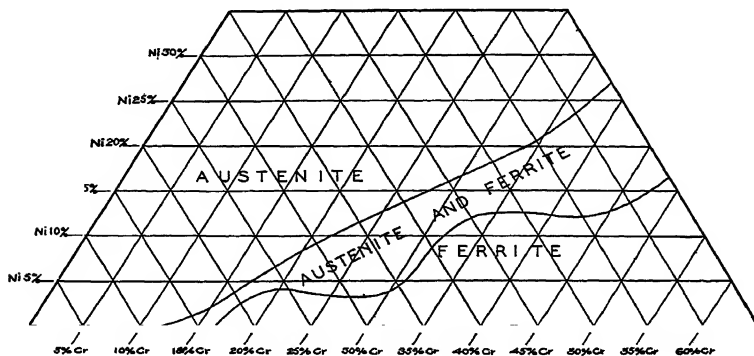


FIG. 35.—PHASE LOCATIONS AT 1300° C.

with substantially constant chromium content. If the actual compositions had really fallen upon straight lines in the triangular diagram, the charts would have been true vertical sections in the three-dimensional phase diagram parallel to the iron-nickel plane. Actually the compositions are not uniformly upon such planes and some very slight discrep-

ancies are introduced which, however, do not detract from the general accuracy of the presentation.

Having thus observed the trends of changes in the resulting structures after quenching from a variety of temperatures, some rational deductions may be made concerning the state of the metal in the system at several temperature levels. Figs. 31 to 35 are, therefore, presented as phase diagrams for the investigated portion of the system at certain designated temperatures.

As has been pointed out, there is some vagueness as to just what alloys can certainly be converted to the *B* constituent. Hence, the ferrite and *B*-constituent regions are drawn overlapping in the long-anneal chart, Fig. 31.

THE HARDENABLE ALLOYS

A reference to the charts, Figs. 15 and 31, will serve to isolate the compositions that are amenable to the development of the hard *B* constituent. It has been found that two characteristic modes of its formation are possible. When a temperature very near the upper limit of its stability is applied, then successively grains of ferrite are recrystallized by an advance of the atomic shift along some crystal planes. This mode does not develop a fine-grain system at all, but on the contrary the grains and grain boundaries remain in situ, certainly an unpredictable phenomenon and one difficult to explain. Only by most careful scrutiny in the microscope can the exact limit of the transformation ordinarily be discovered, but with a heavy scratch made upon the polished and etched surface, the point of advance of the change can be found very easily. Often a band across a single grain marks the advance of the change. It may be inferred that the crystalline nature of the *B* constituent bears some definite relation to the atomic space lattice of ferrite, otherwise a regranulation would occur. However, the authors have not as yet been able to erect the space lattice from ample X-ray films by the Hull method.

Cause of Cracks in B-constituent Alloys

A very large change in volume accompanies this allotropic transformation. It probably amounts to a small percentage. Such shrinkage is not accommodated entirely by plastic flow in these stiff materials with very low ductility. Hence, all specimens of massive *B* constituent prepared in this manner reveal many incipient cracks. The metal is uselessly weak as far as tensile strength properties are concerned and has no shock resistance. Strangely enough, when the *B* constituent is changed again to ferrite by heating, the cracks are self-healing and are closed perfectly, unless they had been opened very widely. Probably the great expansion occurring with reversion to ferrite momentarily presses the surfaces together so intensely as to cause welding or atomic rebonding. At any

rate the surfaces of the cracks would be entirely free from appreciable oxide in compositions of this sort. They oxidize very little with prolonged heating at 1000° C.

The energy content of the *B* constituent is apparently far less than that of the same composition in the ferrite form. Critical-point determinations show substantial heat absorption upon heating and, furthermore, the brittle form is much less attacked in sulfuric acid than is the high-temperature form. The results of critical-point determinations for some of the alloys follow:

CRITICAL POINT FOR *B* TO FERRITE CHANGE FOR SOME CHROMIUM-NICKEL-CARBON ALLOYS

ALLOY No.	CHROMIUM, PER CENT.	NICKEL, PER CENT.	CARBON, PER CENT.	"B" TO FERRITE TEMPERATURE
86	44.6	5.0	0.15	920° C.
88	50.3	10.3	0.25	968° C.
32	41.3	9.3	0.32	964° C.
103	52.1	1.0	0.31	928° C.
108	45.5	10.0	0.19	957° 3., 834° C. (upon cooling)
104	49.8	2.43	0.10	913° C.
25	47.3	0.3	0.44	886° C.
46	50.0	10.1	0.36	961° C.

It will be seen that the greater the nickel content, the higher is the critical point. These critical points however cannot be regarded as well determined in an alloy of this character for there is very great sluggishness exhibited toward any change. An evidence of this fact is the great difference between the location of the point determined upon heating and upon cooling. Alloy 108 was the only one that was active enough to give a cooling point and in this case 123° C. difference exists between the recorded points. With such sluggishness the critical points can only be regarded as indicative of a trend.

The fact that so much shorter time is required to transform the nickel-bearing alloys from ferrite to *B* constituent than to change the nickel-free iron-chromium alloys is explained on the basis of this elevation of transformation temperature. Nickel raises the critical point up to such a temperature that atomic mobility can accomplish restoration to equilibrium. With the almost pure iron-chromium alloys, such temperature as develops the *B* constituent in nickel alloys would be above the critical temperature and therefore have no effect on the destruction of the ferrite.

Cycle of B Constituent on Long Heating

When the sub-critical heating to produce the stable *B* constituent is maintained for a long time at suitably low temperature, the atomic rearrangement does not proceed as described above. Within the ferrite innumerable tiny nuclei of *B* constituent are developed forming a black-

etching material resembling troostite. With this development there ensues marked increase in hardness, usually from approximately 40 to nearly 60 Rockwell *C*. Under a temperature of about 700° C. to 750° C.,

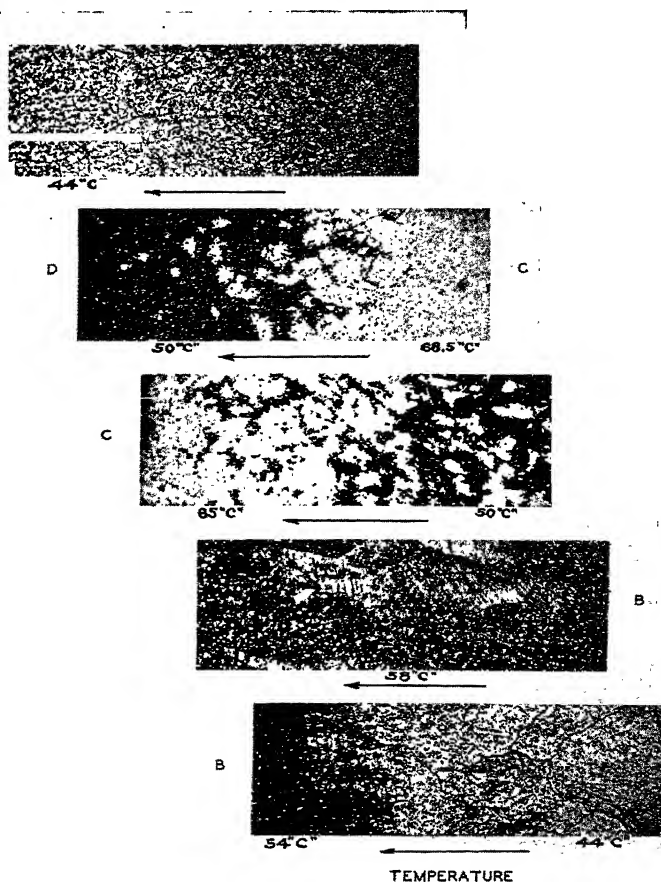


FIG. 36.—SECTIONS ALONG A CONTINUOUSLY HEAT-TREATED BAR OF ALLOY CONTAINING APPROXIMATELY CHROMIUM, 50 PER CENT., NICKEL, 10 PER CENT., AND CARBON, 0.20 PER CENT. $\times 50$.

Each zone was held practically at constant temperature for 3 hr. The temperature increases upward and to the left, the bottom section showing the end held in water and the top section the end held in flame. Rockwell "*C*" hardness is given.

A, original ferrite; *B*, troostitic structure of sub-microscopic *B*-constituent particles; *BC*, supercritical sized particles with some coalesced *B* constituent; *C*, uniform *B* constituent; *D*, restored ferrite at supercritical temperature.

this precipitated *B* constituent does not seem to change substantially, but with higher temperature the particles coalesce and massive *B* is produced which then shows a Rockwell hardness of 68.5 *C*. However, the ever

present cracks in the constituent often render even the estimation of hardness impossible.

Fig. 36 shows the structures found in a torch-treated specimen in which a certain temperature gradient was maintained for 3 hr. One end of a specimen about 5 cm. long was held just under water while a torch flame was directed upon the upper end. Any cross-section of the specimen, therefore, represents the condition developed by holding 3 hr. at some certain temperature between 1000° C. approximately and room temperature. The characteristic structures found are designated by *A*, *B*, *BC*, *C* and *D*. *A* is the original ferrite, *B*, the troostitic structure of sub-microscopic *B*-constituent particles; *BC*, the supercritical sized particles with some coalesced *B* constituent; *C*, the uniform *B* constituent and *D*, the restored ferrite at super-critical temperature.

The lines, so clearly marked in the ferrite, just above the temperature of its formation are probably similar either to *X* bands or Neumann bands and are caused by the sudden expansional stresses developed by volume change. They disappear at still higher temperature. This series shows a complete cycle of the formation and destruction of the *B* constituent.

It is noteworthy that although the precipitated particles in this case gradually coalesce to become the whole of the metal with higher hardening temperature, yet there is slight softening as they grow beyond the critical dispersion size for maximum hardening. Beyond this range of slight softening, the hardness increases very suddenly. Hence it is permissible to regard these alloys as in the class of hardenable alloys exemplified by duralumin and the Sykes alloys.

Effect of Additional Nickel

By the addition of a small percentage of nickel over the amount that permits the entire metal to become *B* constituent, a permanent austenite phase is established which serves to toughen somewhat the alloy in the hard condition. There is no precipitation, of course, in the austenite, but it serves to lessen the tendency toward cracking which often ruins tools made from the alloy. As a cutting tool the hardness is retained very well at high temperature, but there is a certain friability which is not always overcome.

Limiting Compositions for B Constituent

There are probably limiting compositions for the *B* constituent at any temperature. The limits seemingly are narrowed with rise in temperature; certainly the nickel content is increased with rise in temperature. But, what is more important, after a very long time at a dull red heat, part of even the low-chromium alloys (about 30 per cent. chromium) transforms to the *B* constituent. Certain evidences point to a segrega-

tion of chromium into the brittle region. This constituent makes its appearance in the grain boundaries first and thereby renders the alloy quite brittle. Usually the alloy is already very brittle before the extent of the constituent is observable under the microscope and, therefore, this condition is inferred only when more prolonged heating has developed the *B* constituent in measurable amount.

Effect of the B Constituent

An embrittled tube has been photographed by the authors' associate, J. R. Vilella, after several months' maintenance at about 550° C., and the *B* constituent in the dispersed particle condition is plainly visible (Fig. 37). The identity of the *B* constituent was determined by the X-ray spectrometer. Good patterns of the complex crystal of pure *B* constituent had been prepared from various alloys. The brittle tube was fractured and a particularly plane fracture was used as the specimen in the spectrometer. The arrangement of the specimen was such as to aggravate the effectiveness of surface atomic-plane reflection. The fracture surface was of the brittle inter-granular constituent. In addition to a strong ferrite pattern, there were 12 clear supernumerary lines checking perfectly the first 12 strong lines of the *B* constituent.

In some cases then at least, it may be accepted that the slow development of temporary brittleness in chrome-iron is caused by the development of a network of *B* constituent. Heating for several hours at 800° C., or for even 30 min. at 850° C., entirely removes the brittleness by the destruction of the constituent by transformation to ferrite. The removal of brittleness is accompanied by the change in grain boundary appearance shown in the lower views in Fig. 37.

It might well be questioned whether or not the *B* constituent is formed as a result of some usually unconsidered element, such as nitrogen. This question seems to be answered by the behavior of a few heats of metal melted in vacuum with the charge composed of electrolytic iron of exceptional purity and equally fine electrolytic chromium. The carbon content was not possibly more than 0.001 or 0.002 per cent., and it is quite impossible to suspect nitrogen. Such materials develop the *B* constituent with and without nickel.

A more probable explanation lies in the analogy with the gold-copper alloys. Here, according to Johansson and Linde,² at low temperatures with long time heating, a recrystallization sets in and even these most ductile materials become hard and brittle. In this case, also, the X-ray spectrometer answers the question as to the nature of the resultant material.

² C. H. Johansson and J. O. Linde: X-ray Determination of the Atomic Distribution in the Solid Solutions Au-Cu and Pd-Cu. *Annal. Physik* (1925), Pt. IV, 78.

LIMITATIONS OF THE WORK

As has been mentioned, carbon is present in all the alloys studied. It is important to evaluate as far as possible the carbon effect and thereby

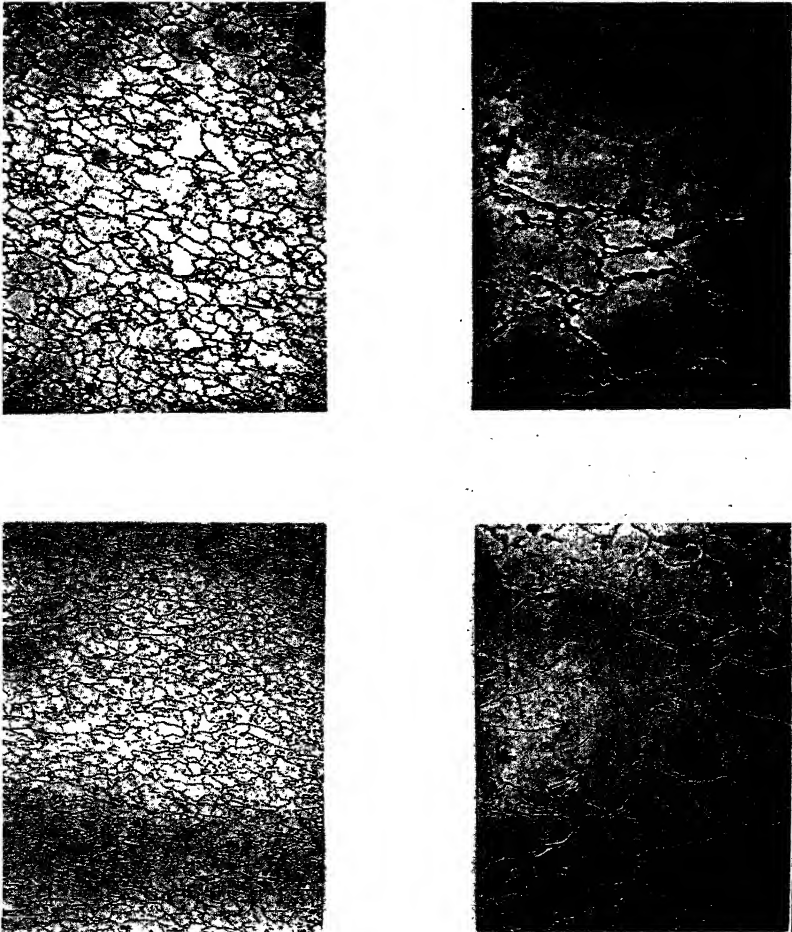


FIG. 37.—STRUCTURES OF EMBRITTLED CHROME-IRON BEFORE (TOP) AND AFTER (BOTTOM) THE RESTORATION OF DUCTILITY BY HEATING. $\times 75$ (LEFT) AND 375 (RIGHT) APPROXIMATELY. SHOWS REMOVAL OF *B* CONSTITUENT.

gain some impression as to the nature of the carbon-free system. Some few facts can be relied on.

1. Carbon resembles nickel in its effect upon the austenitic alloys and, therefore, the boundary of the austenite region is probably somewhat shifted toward the ferrite region by the carbon content.

2. The proportion of austenite in the ferrite-austenite group is increased by carbon.

3. Similarly, alloys that in the absence of carbon would become polyhedral ferrite alone, are rendered martensitic in appearance by the carbon present. Also the magnitude of the martensitic regions in the ferrite-martensite alloys is increased by carbon.

4. In those alloys that are so fundamentally ferritic that they consist wholly of ferrite and carbide particles, the carbon can be neglected.

5. Carbide particles seemingly play no part whatever in the alloys involving the *B* constituent, unless in a very small way. Carbon augments the volume of austenite sometimes associated therewith.

CONCLUSIONS

1. Above about 1000° C. the entire iron-chromium-nickel system is probably composed of only two kinds of solid solutions. One is the gamma-iron-nickel type (austenite) containing some chromium and the other is the alpha-iron-chromium type (ferrite) containing some nickel. These are of the face-centered cubic and body-centered cubic crystal type, respectively. An intermediate zone between the regions of these two pure constituents is composed of the two phases in equilibrium.

2. The ferrite phase increases in composition range with rise in temperature above about 1050° C., due to the influence of high-temperature alpha-iron stability (delta iron).

3. At some temperature above 900° C. (probably below 950° C), heating long enough will develop a new phase, *B* constituent, which is hard (68.5 Rockwell C), brittle and non-magnetic with a complex crystal structure. At this high-temperature level, it appears only in alloys containing about 10 per cent. nickel, but at lower temperatures and still longer heating time, it develops over quite a range of compositions including nickel-free alloys.

4. Many of the alloys containing from 5 to 10 per cent. each of nickel and chromium are martensitic when quenched from a great variety of temperatures and at a great variety of rates. Special melts made in vacuum show that carbon is not necessary for martensite formation, there being a variety of carbonless martensite alloys.

5. The formation of the *B* constituent in small amount in the grain boundaries of the chrome-iron alloys may and does account for some brittleness developed after very long heating at about 550° C.

6. Future work on the Fe-Cr-Ni system should be carried out on carbon-free melts and more properties should be evaluated. Such properties as corrosion resistance and hardness have been investigated by the

present writers. The results cannot be discussed in this brief introduction, but may form the basis for subsequent papers. The similarity as to *B*-constituent formation in the Fe-Cr-Mn and Fe-Cr-CO system has been observed.

The authors desire gratefully to acknowledge their indebtedness to certain of their associates, particularly to John Egan and William Waring, for painstaking assistance in the preparation of the materials for study. Credit for the photomicrography is due J. R. Vilella.

DISCUSSION

J. A. MATHEWS, New York, N. Y.—This paper represents an amazing lot of work, and it is to be regretted it was not done many years ago. This type of steel has been getting attention for at least 15 or 20 years, but this is the first time it has been systematically investigated in connection with all possible variations in composition.

M. A. GROSSMANN, Canton, Ohio.—The most interesting point in the paper is the brittle constituent, which, so far as I am aware, is wholly new and which may be significant of changes to be found in the higher alloy materials.

Mr. Bain and Mr. Griffiths mentioned the possibility that some of the alloys which they show are not really stable but are merely very sluggish, and I think that point must be considered.

There is reference to the position of carbides in the various microstructures. That also seems to be related not only to the solubility of carbon in austenite but also to its solubility in delta iron at high temperature.

There is some reference to the similarity of the effect of nickel and carbon. When carbon is present, nickel appears to have the same effect as more carbon, but there seems to be a question whether, in the complete absence of carbon, nickel might not merely affect the position of the critical point instead of giving various amounts of austenite.

P. A. E. ARMSTRONG, New York, N. Y.—The authors' conclusion that the dark area adjacent to the delta iron is the embrittling constituent in high-chromium irons and steels, with or without nickel, is certainly the result of much painstaking investigation. I regret that evidence was not shown by photomicrographs that the fracture across or in brittle steels, of the analysis investigated, actually occurred through the embrittled structure. Whether or not an embrittling constituent has been located does not affect the fact that certain of these irons or steels, when subjected to high temperature without work, are brittle to shock load.

The fractures of brittle specimens are very similar indeed to low-carbon steels that have been maintained at a black-red or light-blue temperature for a long time. The fracture of such a specimen is around the apparent grain boundaries and the grain appears to be of large size. The large-size grain condition of these high-alloy irons and steels seems to be accompanied by low adhesion between one grain boundary and another; and the grain boundary, instead of being of material width under microscopic observation, is a thin, spidery line.

I realize that the embrittling constituent shown in the authors' photomicrographs was a sizable area occurring in various places and not at all continuous around the grain boundary. I feel that neither the authors nor myself would care to venture the opinion that the microscope has resolved all of the conditions of the structure.

In some high-chromium-nickel irons, and others with no nickel, there is a eutectic-like structure appearing at the grain boundaries in samples that are not as unduly

brittle as those where this structure is entirely absent or only appearing in isolated and apparently unjoined areas.

Whether brittleness should be associated with the formation of delta iron or not, I do not know, although it is fair to say that in some brittle samples delta iron is present.

It is possible to produce in high-chromium irons, which have been rolled or worked at a dull red heat, a very strong tough metal that has an Izod impact value in excess of 100, providing the specimen is notched at 90° to the direction of rolling. Whereas the same piece of alloy steel or iron taken transverse to the direction of rolling and notched in the direction of rolling will have an impact value of 1 or 2 ft.-lb., perhaps less. Delta iron as described by the authors is not discernible in such a sample. The fracture is a mixture of gray, fibrous-looking metal and small, bright, grain-sized, brittle crystals. The specimen does not show one side gray and the other side crystalline, but evenly distributed, sometimes in layers and sometimes not. I recognize that there is a mechanical answer to such a structure and merely refer to this condition as evidence in part that brittleness may be present with delta iron, but brittleness can exist when delta iron apparently does not. The sample referred to, which was subjected to the low-temperature working, was taken from a bar that was of good quality, which had been made by rolling from correct temperatures and represented as good a bar as could be made from this class of alloy. Incidentally, it does not make any difference whether or not such a bar has been annealed.

Another form of brittleness that I have encountered in straight chromium steels and irons, where the carbon is under 0.30 per cent. and the chromium in the middle ranges, say from 8 to 16 per cent. (although it exists outside these ranges), is an enlarged grain structure principally discernible by fracture that occurs in areas across the cross-section of the bar or shape. The fracture appears to be gray and slightly fibrous and near the surface, say perhaps $\frac{1}{2}$ in. from the exterior of the bar, is a ring of about $\frac{1}{8}$ in. wide or an area in the center of the bar or located to one side, not circular or ringlike but just a patch of large bright structure, apparently grain-boundary material, which at that particular spot appears to be very brittle.

Examined under the microscope, under high magnification, this area appears to be of somewhat larger grain size than the surrounding area, while under low magnification there is a defined grouping of grains which appear to have a distinct boundary. This grouping is there under high magnification, of course, but is not so readily seen because the area encompassed is too large for the field. I have seen this type of structure many times in ordinary chrome-nickel structural steels that are used in the automobile trade as drop forgings. Whether the cause for the structure is the same in both types of steel, I do not know. I think that in the high-chromium alloys, this structure is linked with a critical stress area, similar to the well known effects in ordinary iron, with which everyone is familiar.

A sample that is brittle when it has been sharply nicked or notched is not at all brittle when the exterior is continuous and free from areas that can localize the applied stresses. I think the apparent cold shortness that is sometimes present in high-chromium irons and steels, which may have been heat-treated to accentuate such brittleness, is largely a feature of the peculiar type of elongation found in this class of alloy. The tendency is somewhat like that of wrought iron or manganese steels or bronzes to stretch over the entire length of the specimen rather than to immediately neck. Wrought iron is considered a very tough metal and generally speaking is not styled brittle, and yet how brittle it is when nicked in line with rolling. A piece of cold-rolled mild steel is very tough but when nicked it will break off very short, and those who have occasion to use these products have learned by experience how to make use of their valuable properties.

I think the authors in their very diligent research have probably located one of the sources of brittleness but have not, as I see the situation now, found the cause of the

notched brittleness, or cold shortness, that does undoubtedly exist in some of these very valuable alloys. I rather believe that the analysis and condition which surrounds one of the brittle specimens, wherein the embrittling constituent had been located, would not have been brittle at all had this specimen been a strip of 24-gage, and the specimens which were not brittle might easily have shown evidence of being somewhat cold short if they had been 2-in. bars and severely notched.

I realize that my observations are not based altogether on the subject matter of the authors' admirable paper but are more a discussion of a subject arising out of the authors' work and conclusions.

R. L. DOWELL, Minneapolis, Minn. (written discussion).—The authors are to be congratulated on their good work on the iron-chromium-nickel alloys. This kind of research involves much time and study and in many cases is not given the credit it deserves.

There is one point that I wish to discuss in regard to the "B" constituent. The authors have given us a large amount of evidence on the nature of this constituent. Their work shows that its behavior is analogous to that of an intermetallic compound, but they have not assigned a formula to it. It seems that this constituent occurs in the iron-chromium alloys free from nickel. From the atomic concentration standpoint, it seems to correspond to a compound having the formula FeCr. The authors say that when nickel is added to the alloys containing equal proportions of iron and chromium, the critical temperature is raised and the amount of compound gradually decreases. These facts indicate that the compound is of only two components but has a formation temperature which is influenced by the third component (nickel). The authors have pointed out that the formation of this compound takes place from a chrome-ferrite solid solution in about the same manner as the formation of the compounds of the copper-gold system (CuAu and Cu₃Au).

Since this idea came up I have found that Tammann and Treitscke in 1907 pointed out that there were peculiarities in the iron-chromium system which indicated an intermetallic compound (FeCr).

It may be that the hardness of the FeCr (if permissible) may be due partly to a complicated crystal formation caused by the different sized lattice parameters of body-centered iron (2.86 Å) and body-centered chromium (3.65 Å). In the case of the copper-gold compounds, the hardness is not nearly as great.³ This may be due to a less complex crystal lattice being formed because the copper face-centered lattice (3.60 Å) and the gold face-centered lattice (4.08 Å) are considerably closer in their unit parameters.

In conclusion, I wish to ask the authors if there are any reasons why their "B" constituent cannot be called FeCr?

We shall look forward to the publication of the continued studies of the authors.

³N. Kurnakow, S. Zemczuzny and M. Zasedatelev: The Transformations in Alloys of Gold with Copper. *Jnl. Inst. Metals* (1916) 1, 305.

Nature of the Chromium-iron-carbon Diagram

BY MARCUS A. GROSSMANN,* CANTON, OHIO

(New York Meeting, February, 1927)

THIS paper offers for consideration certain somewhat radical modifications in the iron-carbon diagram, these modifications being the result of the presence of notable amounts of alloying elements. When only small amounts of alloys are present, the standard iron-carbon diagram (Fig. 1) can be used with but insignificant changes; when larger amounts of

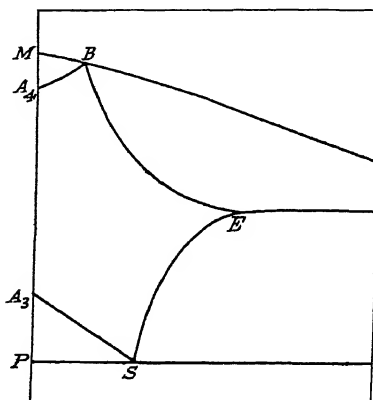


FIG. 1.—IRON-CARBON DIAGRAM.

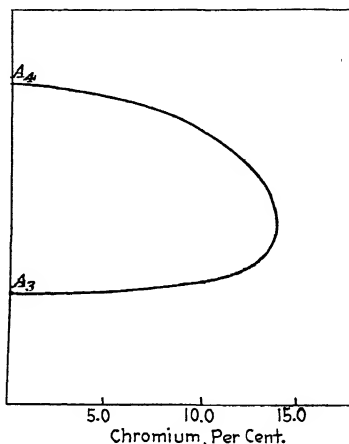


FIG. 2.—IRON-CHROMIUM DIAGRAM.

alloys are present, however, it assumes a distinctly altered appearance. No comment is offered for the present except to call attention to the delta area, A_4BM , which represents a region of mixtures of austenite and delta iron, just as area A_3SP represents mixtures of austenite and alpha iron.

ABSENCE OF CARBON

We may turn at once to the effect of the presence of alloys. It has been found that in the complete absence of carbon, the gamma change will sometimes disappear entirely when sufficient alloying element is

* Chief Metallurgical Engineer, Central Alloy Steel Corp.

present. In the case of chromium, this has been shown by Bain¹ to take place at about 14 per cent. chromium. The equilibrium diagram, as applying to carbonless alloys, is shown in Fig. 2. There is a slight rise in the A_3 point as chromium is added, and a sharp drop in the A_4 point. The two points meet at about 14 per cent. chromium, and beyond that point no transformation to gamma iron takes place. When carbonless iron-chromium alloys, containing more than 14 per cent. chromium, are heated to high temperatures, alpha iron remains as such, being continuous with delta iron, with which it is of course identical.

The same conditions have been shown to obtain with many of the other elements commonly alloyed with iron. Sykes² has shown that in alloys of iron with tungsten, the gamma transformation disappears at 6 per cent. tungsten. He has also shown^{2,3} that it occurs in the case of molybdenum. For silicon Oberhoffer⁴ and Wever⁵ established the disappearance at about 1.85 per cent. silicon. Wever⁵ showed the same for tin at 2 per cent. tin, and Maurer⁶ for vanadium at 2.5 per cent. vanadium. There is evidence that it is true of other alloys as well.

PRESENCE OF CARBON

These conditions are greatly changed, however, when carbon is present. It is well known that many alloy steels containing more than the above limiting percentages can still be hardened in the usual way—that is to say, they are rendered austenitic by heating, and when quenched give martensite. Thus 6 per cent. tungsten steels with 0.65 per cent. carbon (magnet steels) transform and harden in heat treatments much the same as do plain carbon steels. The same is true of 18 per cent. chrome steels with 1 per cent. carbon (stainless steels) and 18 per cent. tungsten steels, with chromium and vanadium, in the presence of 0.7 per cent. carbon (high-speed steels). Steels with 4 per cent. vanadium also harden readily when sufficient carbon is present.

It becomes of importance, then, to determine the relations that exist when various amounts of alloy and various amounts of carbon are present. The evidence offered here deals only with the case where the alloying element is chromium, but it is believed that similar conditions obtain in the case of the other alloys of this type.

¹ E. C. Bain: Nature of Iron-chromium Alloys. *Trans. Am. Soc. Steel Treating*. (Jan., 1926) 9, 9.

² W. P. Sykes: Iron-tungsten System. *Trans.* (1926) 73, 968.

³ W. P. Sykes: Iron-molybdenum System. *Trans. Am. Soc. Steel Treating*. (1926) 10, 839.

⁴ P. Oberhoffer: Zur Kenntnis der Eisen-Silizium Legierungen. *Stahl u. Eisen*. (1924) 44, 979.

⁵ F. Wever: Zur Allotropie des Eisens. *Stahl u. Eisen*. (1925) 45, 1208.

⁶ E. Maurer: Zur Kenntnis der Vanadinstähle. *Stahl u. Eisen*. (1925) 45, 1629.

STUDY OF A "RUSTLESS" IRON

Much significant evidence was developed in a study of an iron-chromium alloy containing, approximately: chromium, 17.0 per cent.; carbon, 0.09 per cent.; manganese, 0.35 per cent.; silicon, 0.60 per cent. This is a material which is being manufactured as a "rustless iron," and it has certain interesting metallographic properties. In the first place it is of the "non-hardening" type—no heat treatment yet devised will raise its hardness beyond about 200 Brinell (the annealed Brinell being about 150). This quality of not hardening is due primarily to the fact that its chromium content is over 14 per cent., and is therefore beyond the limit where carbonless alloys can become austenitic.

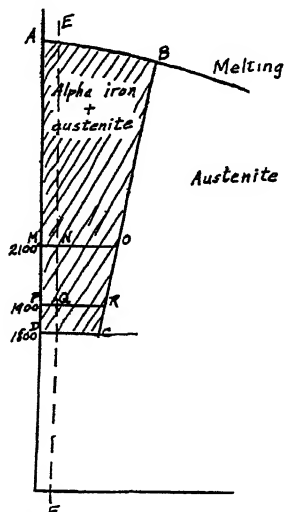


FIG. 3.—EQUILIBRIUM DIAGRAM OF 18 PER CENT. CHROME IRON, SHOWING STRUCTURE AFTER PASSING CRITICAL POINT.

However, it is interesting to observe that although the material cannot be made completely austenitic, it does under certain treatments develop austenite as a certain portion of its volume. There is the apparent "critical point" in the neighborhood of 1800° F. Heating above this temperature results in the presence of austenitic areas, which upon quenching become martensitic. These austenitic areas owe their existence to the presence of the carbides. In spite of the fact that the carbon content is as low as 0.09 per cent., the austenitic areas form an appreciable and definite proportion of the structure.

CONSTITUTION BEYOND THE "CRITICAL POINT"

Moreover, it is significant of the constitution of these alloys that as the temperature is raised further and further beyond 1800° F., the pro-

portion of austenite decreases continuously, the decrease continuing up to the melting point. Attention was directed to this fact in an earlier paper.⁷ After 6 hr. heating at 1900° F., the austenite was about 35 per cent. of the total volume. After a similar heating at 2100° F., the proportion of austenite was only about 25 per cent. A piece of such material cut from an ingot (*i. e.*, it represented conditions at the temperature of solidification) showed a proportion markedly lower than even 25 per cent. If we attempt to draw an equilibrium diagram which would indicate these phenomena, we should have to draw it as in Fig. 3. Ordinates as usual indicate temperature and abscissae the carbon content. The line *EF* represents the alloy under consideration. The area *ABCD* indicates a region of mixed structures, alpha iron and austenite. When the critical point at 1800° F. is passed, the structure becomes a mixture of alpha iron and austenite. As the temperature is raised, the proportion of austenite decreases, as indicated by the slope of the line *BC* and determined in the usual way by the distances of points on the lines *AD* and *BC* from *EF*. Thus at 1900° F. the length of *PQ* (representing amount of austenite) is about 35 per cent. of the length *PR*. At 2100° F., *MN* is about 25 per cent. of *MO*.

It became necessary next to seek the origin of a diagram of the type shown in Fig. 3. This was forthcoming in a series of high-temperature heating experiments. Alloys low in carbon, with graduated alloy contents, were heated at a series of successively higher temperatures, and quenched in water. The high-temperature structures were thus preserved. (Due to the slowness of diffusion of the constituents in high-chromium materials, it is possible generally to preserve structures quite undisturbed if the cooling is done with moderate rapidity.)

STRUCTURE VS. ALLOY CONTENT AND TEMPERATURE

Fig. 4 shows the material with the lowest alloy content. Here the first and second heating temperatures gave an austenitic structure, but at the highest temperature delta iron is beginning to appear in the grain boundaries. Fig. 5 shows the next higher alloy content. The lowest quenching temperature gives only austenite. At the intermediate quenching temperature, delta iron has appeared. At the highest quenching temperature, it is important to note that the proportion of delta iron has increased. Fig. 6, with still more alloy, shows a continuation of the same tendency, in more marked degree. Delta iron has appeared at the first temperature, and has increased in amount definitely as the temperature has been raised.

⁷ M. A. Grossmann: Behavior of Carbon in a High-chromium Rustless Iron. *Trans. Am. Soc. Steel Treating*. (Sept., 1926) 10, 436.

INDICATION OF STRUCTURAL CHANGES IN EQUILIBRIUM DIAGRAM

If we attempt to indicate these changes in an equilibrium diagram, we may to advantage revert first to the iron-carbon diagram of Fig. 1.

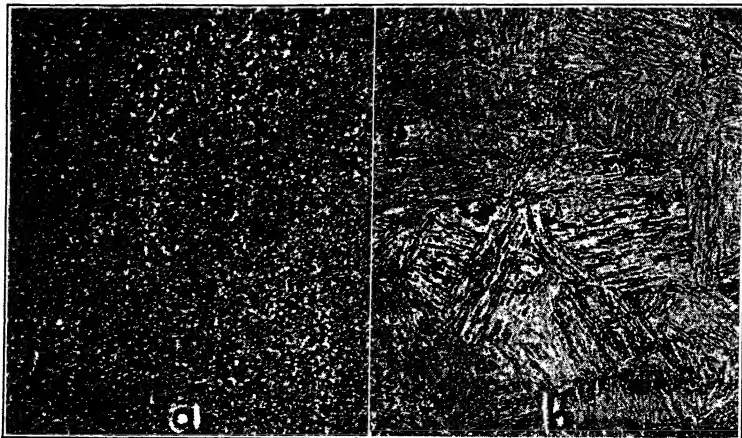


FIG. 4a.

FIG. 4b.

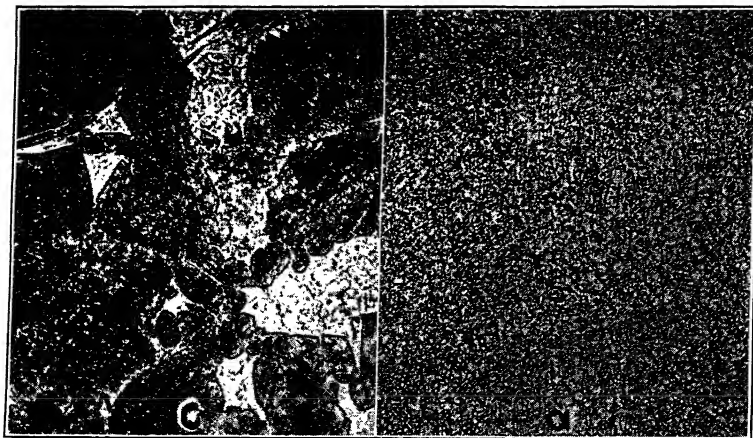


FIG. 4c.

FIG. 5a.

FIG. 4.—MATERIAL WITH LOWEST ALLOY CONTENT. $\times 250$. (a) 1600° F., (b) 2000° F., (c) 2400° F. (a) AND (b) SHOW AUSTENITIC STRUCTURE; IN (c) DELTA IRON IS BEGINNING TO APPEAR IN GRAIN BOUNDARIES.

FIG. 5.—MATERIAL WITH NEXT HIGHER ALLOY CONTENT. $\times 250$. (a) 1600° F.

Attention is directed to the delta iron region, A_4BM , because it will be necessary to determine its relation to the alloys under investigation. Figs. 4, 5 and 6 exhibit features of profound significance. It is observed

at once that, with every increase in alloy content, there is a lowering of the temperature at which delta iron first appears. Of equal importance is the obvious fact that as the temperature is raised, the proportion of delta iron increases, *i. e. the proportion of austenite decreases*. This leads to the present assertion that the delta iron phenomena which occur in alloy materials can be represented in a diagram quite similar to Honda's area A_4BM , with the revision that the line A_4B is gradually lowered as the alloy content is increased.

The area A_4BM is to be interpreted as a region of mixed structures, of delta iron and austenite. When an alloy such as EF in Fig. 7 is heated to point x , it just enters the delta region, and there will be present

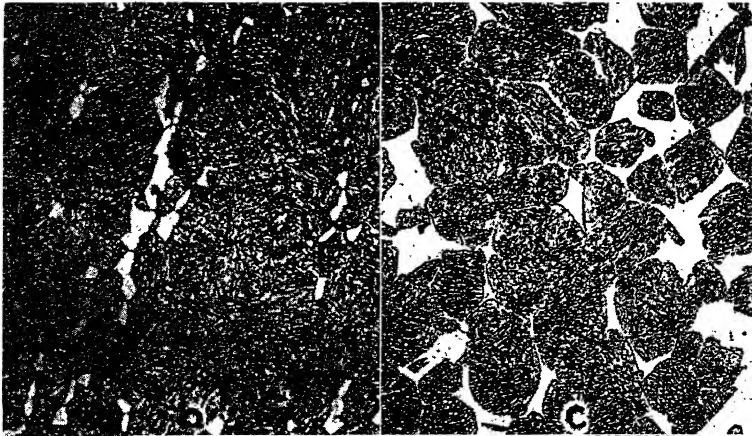


FIG. 5.—(CONTINUED.) (b) 2000° F., (c) 2400° F. (a) SHOWS ONLY AUSTENITE; IN (b), DELTA IRON IS DISCERNIBLE; (c) SHOWS INCREASED PROPORTION OF DELTA IRON.

but a small proportion of delta iron. When the temperature is raised to y , the proportion of delta iron increases and the proportion of austenite decreases, as represented by the slope of the line A_4B . With still further rise of temperature, to z , the proportion of austenite decreases still further. That these same phenomena occur in the alloyed materials is indicated clearly in Figs. 5 and 6. We see there the increase in delta iron and the corresponding decrease in the amount of austenite. The phenomena recorded in Figs. 4, 5 and 6 could therefore be represented by the lines A_4B in the three diagrams of Fig. 8. The carbon content of the particular alloys is indicated by the line EF which represents the alloy. Its entrance into the delta region is accompanied by the formation of certain definite proportions of delta iron and austenite, as indicated by the position of the line A_4B .

TERNARY DIAGRAM OF IRON, CHROMIUM AND CARBON

We are ready now to proceed to the generalization on the nature of the ternary diagram of iron, chromium and carbon. This generaliza-

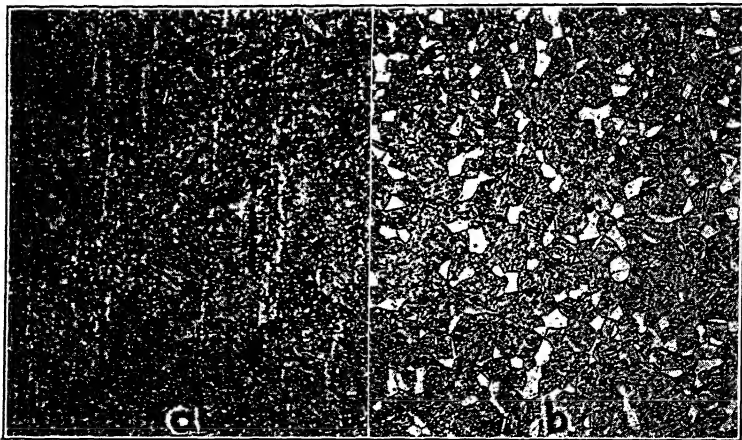


FIG. 6a.

FIG. 6b.



FIG. 6c.

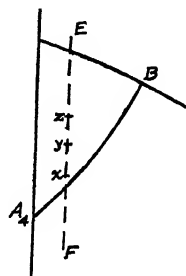


FIG. 7.

FIG. 6.—MATERIAL CONTAINING HIGH ALLOY CONTENT. $\times 250$. (a) 1600° F., (b) 2000° F., (c) 2400° F. DELTA IRON PRESENT AT (a) AND IN INCREASING PROPORTION IN (b) AND (c).

FIG. 7.—DELTA REGION IN IRON-CARBON DIAGRAM.

tion will show the structures in the materials with lower alloy content, and will permit an extension into the region where the alloy content is

high enough to cause decided changes. A ternary diagram would seem to be clearer in three dimensions than in the customary plane triangle, hence an attempt will be made to present it so here. Instead of a tri-

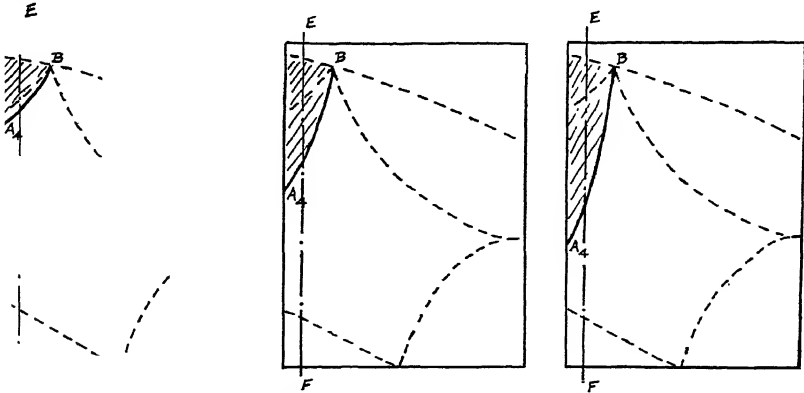


FIG. 8.—IRON-CARBON DIAGRAMS SHOWING CHANGE IN POSITION OF DELTA REGION WITH RISE IN ALLOY CONTENT.

angular prism, it has been decided to use rectangular coordinates, plotting percentage carbon in one horizontal direction, percentage chromium in

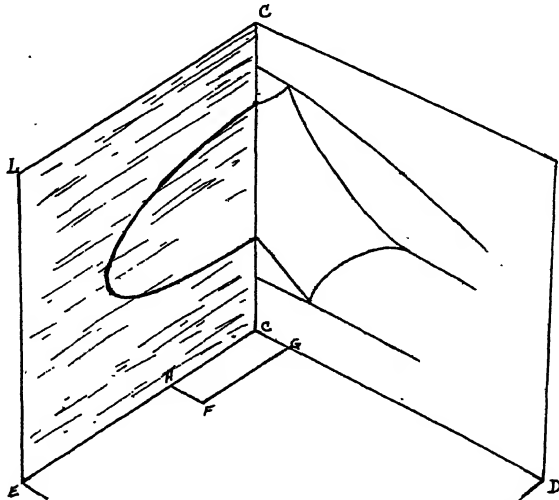


FIG. 9.—TERNARY DIAGRAM OF IRON, CHROMIUM AND STEEL.

the other horizontal direction and temperatures vertically. The zero ordinate is of course pure iron, as shown in Fig. 9. Such a structure presents the elements of a true ternary diagram, with percentage of carbon

parallel to CD , percentage of chromium parallel to CE , and temperature indicated vertically parallel to CC' . Thus a point F would represent an alloy with GF per cent. chromium and HF per cent. carbon, and in this same way all the different combinations of chromium and carbon can be represented within the volume indicated in the diagram. It will be observed that the right-hand boundary plane $C'RD$ represents iron-carbon alloys, without chromium. On it can be plotted the iron-carbon diagram of Fig. 1, indicating the various temperatures of transformation. The left-hand boundary plane $C'LE$ represents the series of iron-chromium alloys, free of carbon. On this can be plotted the iron-chromium diagram

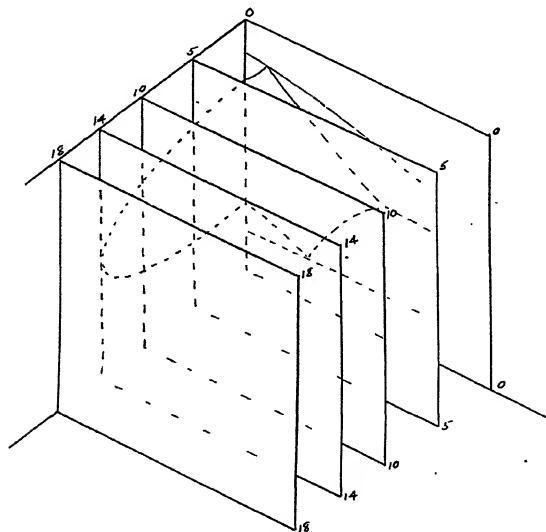


FIG. 10.—METHOD OF CONSTRUCTING THREE-DIMENSIONAL DIAGRAM.

of Fig. 2, showing the transformations in carbonless iron-chromium alloys. It is proposed to fill in the central portion, indicating the transformations in alloys having various proportions of iron, chromium and carbon.

NATURE OF THE TERNARY DIAGRAM

It has been found convenient in the course of the present work to show the nature of the ternary diagram by taking a series of plane sections through it. The sections are vertical and are parallel, and are also parallel to the original (chrome-free) iron-carbon diagram. We have thus, in effect, a series of iron-carbon diagrams, and it will be found that the familiar fields of the standard diagram are present, but are modified by the presence of the alloy. Fig. 10 shows how the sectional planes are placed in the ternary diagram of Fig. 9. The plane 0-0-0 is the standard

iron-carbon diagram. The plane 5-5-5 is a new iron-carbon diagram, modified by the presence of 5 per cent. chromium—that is, the standard iron-carbon diagram has been moved out in the direction of increasing

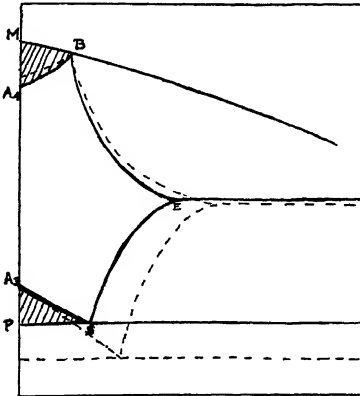


FIG. 11.

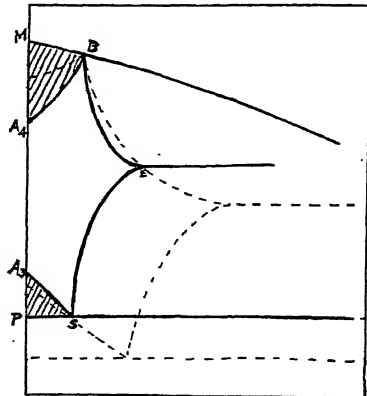


FIG. 12.

FIG. 11.—FIVE PER CENT. CHROME ALLOY. (PLANE 5-5-5 IN FIG. 10.) DOTTED LINES REPRESENT STANDARD (CHROME-FREE) IRON-CARBON DIAGRAM.

FIG. 12.—TEN PER CENT. CHROME ALLOY. (PLANE 10-10-10 IN FIG. 10.)

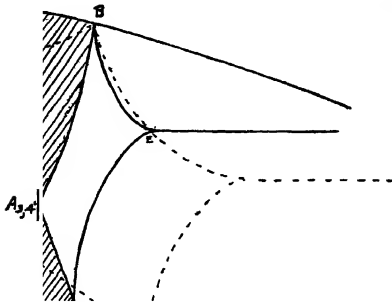


FIG. 13.—FOURTEEN PER CENT. CHROME ALLOY. (PLANE 14-14-14 IN FIG. 10.)

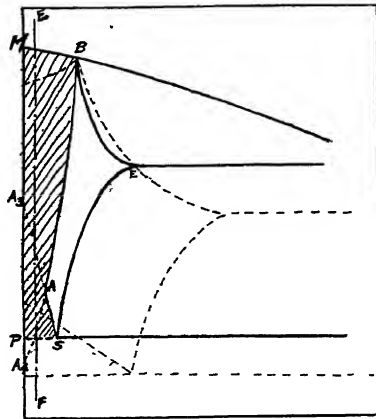


FIG. 14.—EIGHTEEN PER CENT. CHROME ALLOY. (PLANE 18-18-18 IN FIG. 10.)

chromium, by an amount corresponding to 5 per cent. alloy. This procedure would seem to be logical, and is justified by the fact that (as will presently appear) the familiar structures of the iron-carbon diagram are found quite similarly in the alloys with chromium. The presence of the chromium merely changes the position of the lines in the alloy-free diagram, such as the lines A_4B , A_3S and SE . It has been found in this work

that the diagrams can thus be used to present clearly the structures found in the various alloys, and to show their relations to the others of the group. We may conceive, then, a series of iron-carbon diagrams, with increasing alloy content, as illustrated in Fig. 10. Section 10-10-10 is at 10 per cent. chromium, section 14-14-14 is at 14 per cent. chromium, and section 18-18-18 is at 18 per cent. chromium.

ALLOYS CONTAINING VARIOUS PERCENTAGES OF CHROMIUM

If we revert to Figs. 4, 5 and 6, we observe the changes which have already been summarized in the diagrams of Fig. 8, giving the change of position of the line A_4B . Reference should now be made to Figs. 11, 12 and 13, which represent the planes 5-5-5, 10-10-10 and 14-14-14 in the

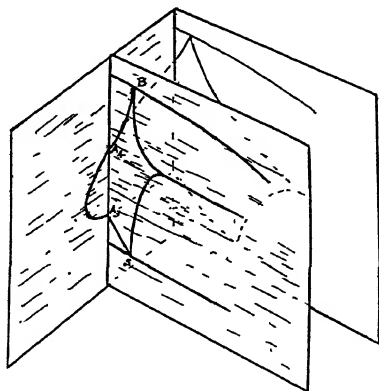


FIG. 15.—IRON-CARBON DIAGRAM WITH 10 PER CENT. CHROMIUM, IN POSITION IN TERNARY DIAGRAM.

diagram of Fig. 10. We must remember first that the point A_4 (carbon content zero) follows the course indicated in Fig. 2. Since Fig. 2 is now the left-hand boundary plane of the ternary diagram, the line A_4B of the iron-carbon diagrams will always be found sloping up from the top of the gamma iron "loop," as illustrated in Fig. 15. Fig. 15 indicates the position of plane 10-10-10 in the three-dimensional system, *i. e.* it shows the diagram of Fig. 12 in its proper position. This relation (of the line A_4B to the upper boundary of the diagram of Fig. 2) holds for all alloys from zero to 14 per cent. chromium. At the same time the point A_3 rises, in a similar manner, along the lower boundary of the diagram of Fig. 2. The line A_3S is shown likewise in Fig. 15, its position being determined partly by the change of A_3 as indicated. It is to be observed, then, that as the alloy content is increased, the lines A_4B and A_3S approach one another. When the alloy content reaches 14 per cent., the two lines meet, *i. e.* they intersect at the point $A_{3,4}$, as shown in Fig. 13. The iron-carbon diagram with this alloy content is just at the end of

the gamma iron loop; at this point with carbon-free alloys the change to gamma iron just disappears. It is the plane 14-14-14 in Fig. 10.

The positions shown for the line SE in these diagrams are those established by Monypenny.⁸ The variations in the temperature of the A_1 point are taken from the work of Edwards, Sutton and Oishi.⁹ It is of interest to observe that as the chromium content increases, the carbon content of the eutectoid composition becomes lower.

MERGING OF ALPHA IRON AND AUSTENITE AREAS

We must return now to the diagram of Fig. 3, which shows the structures in alloys containing 18 per cent. chromium. Consideration of Figs. 11, 12 and 13 permits an extension to these higher alloy contents, as shown in Fig. 14. In Fig. 11, the cross-hatched area A_4BM represents a mixture of delta iron and austenite. Similarly, the cross-hatched area A_3SP represents a mixture of alpha iron and austenite. Since alpha iron and delta iron are identical in structure, it is important to note that these two cross-hatched areas represent fields of entirely similar structures. When the amount of alloy is increased, as in Fig. 12 (10 per cent. chrome), the point A_4 drops and the point A_3 rises, so that the areas A_4BM and A_3SP approach each other. At 14 per cent. chrome (Fig. 13) the points A_4 and A_3 meet at the point $A_{3,4}$ —that is, the point A_4 has continued its drop, and the point A_3 has continued its rise, until now the two points have met as shown. The areas A_4BM and A_3SP , representing entirely similar structures, have just met at the point $A_{3,4}$. With still higher alloy contents, the two areas continue their tendency to approach, and join to form a united area, as $BASPM$, Fig. 14. We can imagine that the line BA_4 has continued its drop, and the line SA_3 its rise, until the two lines (instead of merely meeting) have actually crossed, as at A in Fig. 14. The points A_4 and A_3 then become imaginary, as indicated by the dotted portions AA_4 and AA_3 of the lines BA_4 and SA_3 in Fig. 14. We have then, in the region $PSABM$, a mixture of alpha iron and austenite throughout.

DECREASE OF AUSTENITE WITH INCREASE OF TEMPERATURE

Consider an alloy of the carbon content represented by the line EF . If such an alloy is heated, it enters at some temperature a region of mixed alpha iron and austenite. As it is heated further, its structure still remains a mixture of alpha iron and austenite, except that at the higher temperatures the amount of austenite decreases as indicated by the slope of the line AB . This recalls the problem presented in Fig. 3,

⁸ J. H. G. Monypenny: Structure of Some Chromium Steels. *Jnl. Iron Steel Inst.* (1920) I, 493.

⁹ C. A. Edwards, H. Sutton and G. Oishi: Properties of Iron-chromium-carbon Steels. *Jnl. Iron Steel Inst.* (1920) I, 403.

and it will be observed that Fig. 14 represents the solution of that problem. A comparison of the area *DCBA* of Fig. 3 with area *PSABM* of Fig. 14 shows that they represent the same kinds of structures, and there

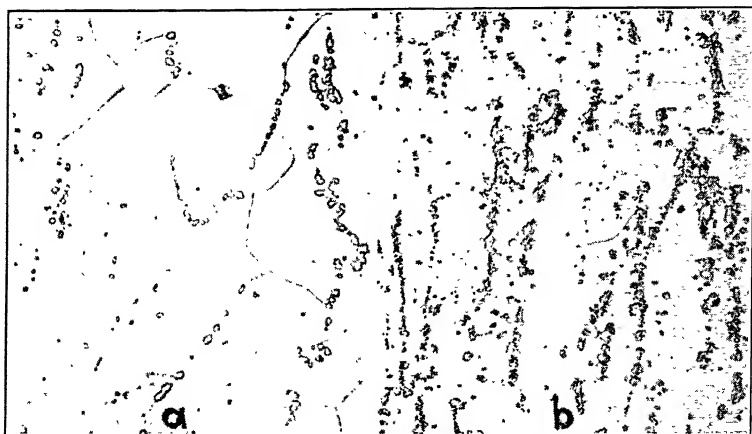


FIG. 16a.

FIG. 16b.

FIG. 16.—SLOW-HEATING SERIES SHOWING GROWTH OF AUSTENITIC AREAS FROM CARBIDE PARTICLES. (a) Two hours at 1350° F. $\times 500$. (b) Two hours at 1450° F. $\times 500$.

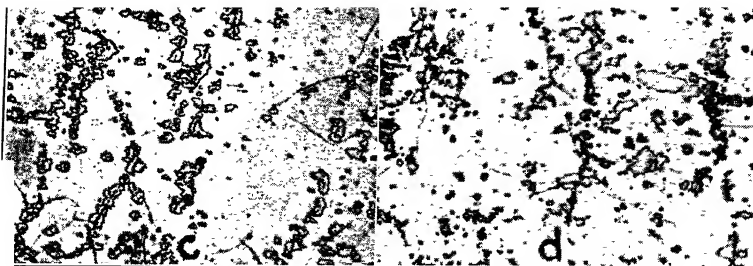


FIG. 16c.

FIG. 16d.

FIG. 16.—(CONTINUED.) (c) Two hours at 1550° F. $\times 500$. (d) Two hours at 1650° F. $\times 500$.

would seem to be no doubt of their origin. Such an area seems to be typical of the structures to be obtained with low-carbon materials containing 14 to 20 per cent. chromium.

EXPLANATION OF "CRITICAL POINT"

The apparent "critical point" at 1800° F. requires some explanation. It will be observed that the A_{c1} point in high-carbon alloys with the same



FIG. 16e.

FIG. 17a.

FIG. 16.—(CONTINUED.) (e) Two hours at 1750° F. $\times 500$.

FIG. 17.—SERIES OF SUCCESSIVELY HIGHER HEATINGS OF MATERIAL COMPOSED OF: CARBON, 0.13 PER CENT.; SILICON, 0.35 PER CENT.; CHROMIUM, 14.45 PER CENT. ALPHA IRON INCREASES IN AMOUNT AS QUENCHING TEMPERATURE IS RAISED. (a) 1600° F. $\times 250$.



FIG. 17b.

FIG. 17c.

FIG. 17.—(CONTINUED.) (b) 1800° F. $\times 250$. (c) 2000° F. $\times 250$.

chrome content (17 per cent.) is at about 1500° F. It seemed entirely probable, therefore, that if any true critical point occurred in the low-

carbon alloys, it would be found also in the neighborhood of 1500° F. The structure of the alloy affords an opportunity for discovering the nature of the anomaly. It must be remembered that the material contains 18 per cent. chromium and that, when the carbon is precipitated as carbide, the matrix in which the carbide is embedded is really a *carbonless* 18 per cent. chrome-iron alloy. This percentage of chromium is in the range where carbonless alloys form no gamma iron on heating, and the capacity of alpha iron (even when heated) to dissolve carbon is probably negligible. There would therefore be no formation of gamma iron or austenitic areas in the carbonless matrix. But it is quite conceivable that a slow diffusion of carbon might begin from the boundaries

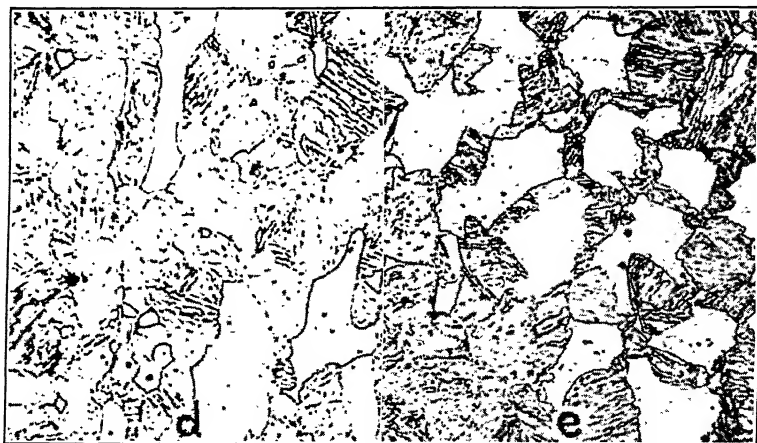


FIG. 17.—(CONTINUED.) (d) 2200° F. $\times 250$. (e) 2400° F. $\times 250$.

of the carbide particles themselves. The boundary layers between the carbide particles and the matrix would not be completely crystallized, and might in addition be impoverished in chromium by precipitation of chromium in the carbide. The formation of austenitic areas would under these circumstances be a resultant of time and temperature, and it would be possible to trace the growth of the austenitic areas from the carbide particles, by slow progression into areas of gradually increasing size.

FORMATION OF AUSTENITE AT SLOW RATES OF HEATING

A slow-heating series, carried out to test this point, gave good confirmatory evidence. Fig. 16 shows the structure of pieces heated for 2 hr. at temperatures beginning at about 1400° F. The pieces were all heated together, so that the higher temperatures represent cumulative heatings which include the times at all the lower temperatures. It is

seen at once that small austenitic areas begin to form at about 1500° F., and that these grow progressively larger at higher temperatures. It is also significant that the proportion of austenite which is formed, with

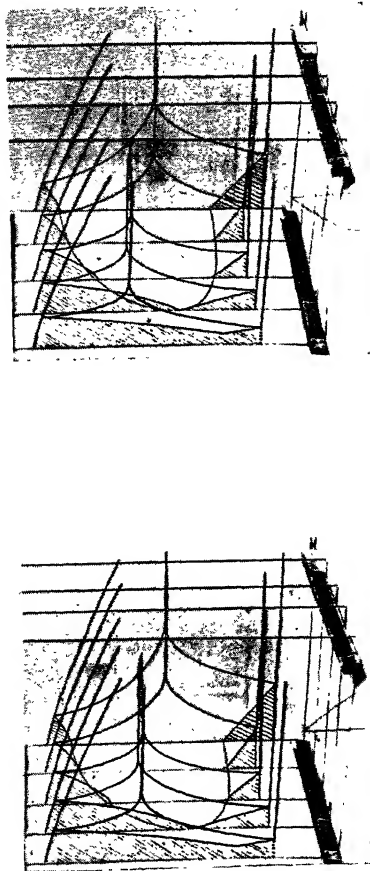


FIG. 8.—CHROMIUM-IRON-CARBON DIAGRAM.
Stereoscopic photograph of three-dimensional equilibrium diagram. Illustration should be removed from page and viewed in a stereoscope. It is possible also (with a little practice) to see it stereoscopically without the instrument.

ordinary rates of heating, at about 1800° F., is at these slow rates of heating found between 1600 and 1700° F. A still slower rate of heating would undoubtedly find the full proportion at even lower temperatures. It seems entirely probable that with a very long heating the austenite

would form in its proper proportion at say 1525° F., thus corresponding with the usual and expected critical points in the other alloys. The "critical point" at 1800° F. merely meant that, at the comparatively rapid rate of heating customarily employed, a temperature of 1800° F. was reached before appreciable amounts of austenite were formed.

It was thought to be of interest also to include photographs of a low-carbon material in which the continuous persistence of alpha-delta iron is just barely present, its amount at the lowest temperatures being very small. Fig. 17 shows a series of successively higher heatings of a material of the following composition: carbon, 0.13 per cent.; silicon, 0.35 per cent.; chromium, 14.45 per cent. The amount of alpha iron (white areas) is very small at the lowest temperature, and gradually increases in amount as the quenching temperature is raised.

EFFECT OF OTHER ALLOYS ON CHROME IONS

Finally, it remains to be stated that other elements which form the gamma iron loop will, when present with the chromium, act to depress the delta region in the same way that additional chromium would depress it. Figs. 4, 5 and 6, while entirely similar to plain chromium alloys, are in reality photographs of a series of alloys all containing about 12 per cent. chromium but with silicon contents of 1, 2 and 3 per cent. It will be recalled that Oberhoffer¹⁰ and Wever¹¹ report the existence of the gamma iron loop in iron-silicon alloys. Evidence is at hand that other alloys act in the same way.

It is recommended that the assembled chromium-iron-carbon diagram, shown stereoscopically in Fig. 18, be removed from the page and viewed in a stereoscope.

ACKNOWLEDGMENTS

The author wishes to acknowledge the assistance of his associates, J. D. Gat, C. C. Snyder and P. E. Macklin in the preparation of illustrations.

ADDENDUM

Further evidence has become available regarding the reactions in the delta region. In Figs. 1, 7, etc., the line A_4B was drawn solely to indicate temperatures at which delta iron makes its appearance on heating. There was no reference to other reactions in the delta region. Theoretical considerations would have demanded drawing the region A_4BM as shown in Fig. 19. The area A_4CMA_4 in Fig. 19 indicates a region of pure delta iron, without austenite. This presupposes that whatever carbon is present is soluble in the delta iron in that region. The area A_4CBA_4 is a region of mixed structures, delta iron plus austen-

¹⁰ P. Oberhoffer: *Op. cit.*

¹¹ F. Weaver: *Op. cit.*

ite. Point *B* is a "triple point," indicating the simultaneous presence of delta iron, austenite and liquid melt.

The additional evidence has to do with the region in which delta iron is present alone, without austenite. It is evident that the boundary of this region, the line *A₄C*, represents the saturation solubility of carbon in delta iron, at the respective temperatures. Whereas this solubility was previously indeterminate, it appears now that it can be determined experimentally. The method involves determining, with each alloy, the temperature at which the alloy when heated emerges from a region of mixed structures (delta iron plus austenite) into a region of pure delta iron.

As stated before, the high-temperature structures can be preserved to room temperature fairly well by quenching the test pieces, because

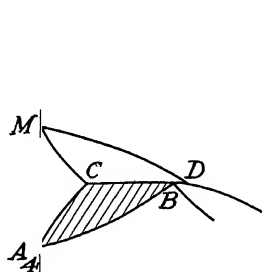


FIG. 19.—DELTA REGION.

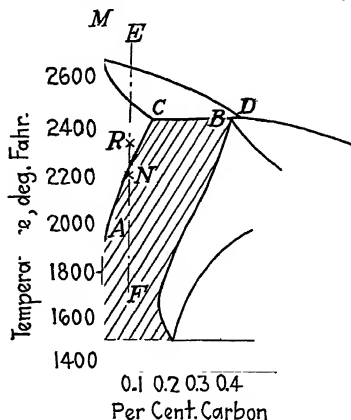


FIG. 20.—IRON-CARBON DIAGRAM WITH 18 PER CENT. CHROMIUM.

the reactions are very sluggish. In the present case, favorable test conditions were obtained by putting the alloy in the form of thin strip, which was quenched in water from the high temperature. The reactions are typified in the 18 per cent. chrome alloy mentioned previously, containing 0.09 per cent. carbon. The structures found in this alloy are summarized in the constitution diagram of Fig. 20, the alloy named being represented by the line *EF*. Up to temperatures of nearly 2200° F., corresponding, say, to point *N*, Fig. 20, the microstructure obtained after quenching is about as shown in Fig. 21. It shows a mixture of alpha iron and austenite (martensite). When the temperature has passed 2200° F., as at point *R*, Fig. 20, the structure of Fig. 22 is presented. Here the original austenite grains have disappeared. The structure at the high temperature consisted solely of delta iron. It is true that austenitic areas are present, but these have resulted from

rejection of carbon (and austenite) to the grain boundaries of the delta iron in cooling. The cooling was not quite rapid enough to prevent formation of austenite.

This appearance of austenite at the grain boundaries may be prevented by quenching from somewhat higher temperatures. When

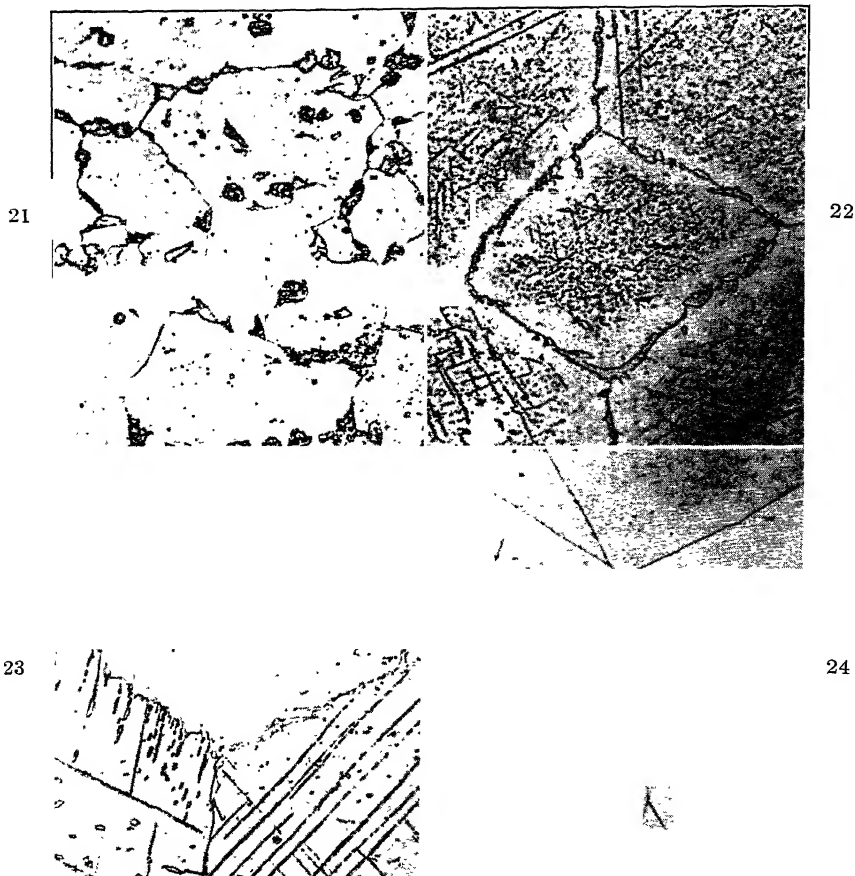


FIG. 21.—A CHROME IRON QUENCHED FROM JUST BELOW 2200° F.

FIG. 22.—SAME CHROME IRON AS IN FIG. 21, QUENCHED FROM JUST ABOVE 2200° F.

FIG. 23.—SAME CHROME IRON AS IN FIG. 21, QUENCHED FROM JUST ABOVE 2300° F.

FIG. 24.—SAME CHROME IRON AS IN FIG. 21, QUENCHED FROM JUST ABOVE 2400° F.

the temperature previous to quenching is a little higher (say 2300° F.) there is not sufficient time for migration of the carbon (and austenite) to the grain boundaries. However, some austenite still makes its appearance in the cleavage planes of the delta iron, as shown in Fig. 23.

The structure is quite similar to the familiar Widmanstätten structure, and its mode of formation is analogous. If the alloy has been heated to still higher temperature (say 2400° F.) before quenching, then the rate of passing through the critical zones is so rapid that no austenite whatever forms on cooling, and the structure consists solely of delta iron (Fig. 24). The carbon present precipitates as carbide particles, since it is no longer soluble at room temperatures.

It thus appears that the original austenite areas disappeared at about 2200° F. in this particular alloy, *i. e.* at that temperature it became wholly delta iron. In the same way all points on the line *AC* of Fig. 20 may be determined by finding where the original austenite (of the mixed structures) disappears.

Without following the changes through all the various alloy contents, it will suffice to say that the evidence indicates a progressive change from the structures of the chrome-free alloys of Fig. 19, to those of the alloys with about 18 per cent. chrome, as shown in Fig. 20.

It should be added that structures similar to those of Fig. 6c demonstrate the presence of the three structures delta iron, austenite and melt simultaneously, as represented on the line for triple structure *CBD* of Fig. 20.

DISCUSSION

E. C. BAIN, Long Island City, N. Y.—Mr. Grossmann's painstaking and enlightening researches point strongly to a conclusion that a very considerable amount of carbon is soluble in delta iron. Since delta iron is alpha iron at elevated temperature, we may say that the solubility of carbon in ferrite increases greatly with elevation of temperature.

This circumstance gives us no choice but to show such solubility in a diagram. By comparison with other alloys the amount is relatively small, and such solubility differs from carbon solubility in ordinary alpha iron only in degree and not in kind. Since the solubility of carbon in alpha iron is ordinarily not shown in iron-carbon diagrams we are confronted with the problem of possible scientific inconsistency. Mr. Grossmann shows that he is well aware of this fact and has, perhaps, treated the discovery in the best possible manner.

The circumstance brings to mind Dr. Guertler's insistence upon indicating an almost general slight solubility in his equilibrium diagrams by means of dotted lines close to the 0 and 100 per cent. boundaries of his diagrams. Being dotted lines, the exact composition is not fixed and yet the requirements of correct principle are satisfied.

Mr. Grossmann's determinations show as high as 0.15 per cent. carbon dissolved in delta iron. My own studies indicated somewhat higher values but I believe Mr. Grossmann's figures are more accurately estimated and therefore more reliable than my own. It is to be hoped that further studies of this nature will be forthcoming and that the present inaccurate conceptions of the iron-chromium-carbon system will be replaced by better information.

A Comparison of the Effect of Nickel and Cobalt in Steel

BY FRANKLIN H. ALLISON, JR., PITTSBURGH, PA.

(New York Meeting, February, 1927)

THE influence of cobalt and nickel on the properties of steel might be readily expected to be very similar. The two elements occupy close and somewhat unusual positions in the periodic table, their chemical behavior and compounds are so completely analogous and their various physical properties, including the ferro-magnetic characteristic, show such a remarkable correspondence, that there is no doubt of a strong resemblance existing between these two metals. However, the variety of properties that are obtained from steel is due largely to the fact that the element iron possesses allotropic forms. Alloying elements that affect these allotropic forms, or alter the nature of the allotropic changes, exert a profound influence on the resulting steel. In the case of nickel and cobalt, although these elements are very similar chemically and physically, it is found that they differ distinctly in their individual effects on the critical points of iron.

Both nickel and cobalt form a series of homogeneous solid solutions with iron at temperatures above 1800° F. The difference in their behavior does not occur until changes begin to take place in the solid state below this temperature. An inspection of the respective thermal equilibrium diagrams at the lower temperatures makes the opposite effects of nickel and cobalt clearly evident.

EFFECT OF NICKEL AND COBALT IN BINARY ALLOYS WITH IRON

The binary diagram of iron-nickel has been investigated by a number of workers. The lower temperature portion in Fig. 1 is reconstructed from the data of D. and H. Hanson.¹ The diagram of iron-cobalt, shown in Fig. 2, is taken from the work of Ruer and Kaneko.² Masumoto³ has recently discovered a lower critical point in cobalt and this has been added to the diagram. In both diagrams the space lattice structures as determined by Andrews⁴ have been included. It is interesting to note

¹ D. Hanson and H. Hanson: The Constitution of the Nickel-iron Alloys. *Jnl. Iron Steel Inst.* (1920, No. II) 102, 39.

² R. Ruer and K. Kaneko: Das System Eisen-Kobalt. *Ferrum* (1913) 11, 33.

³ H. Masumoto: *Sci. Repts. Tohoku Imp. Univ.* No. LV, 15, 449.

⁴ M. Andrews: X-ray Analyses of Three Series Alloys. *Phys. Rev.* (1921) 17, 259.

that the body-centered cubic structure of alpha iron disappears entirely with concentrations of nickel greater than 32 per cent., whereas this same structure persists with cobalt additions until concentrations of more than 80 per cent. have been reached.

In referring to the iron-nickel diagram, it is readily apparent that additions of nickel depress the temperature of the alpha-gamma change with an extreme rapidity. The contrary effect occurs in the iron-cobalt diagram where the cobalt additions produce a gradual elevation of the A3 temperature. The magnetic change in both diagrams is inflected in the direction of the allotropic change and soon becomes coincident with it. A most important difference in the two figures, however, is that in the case

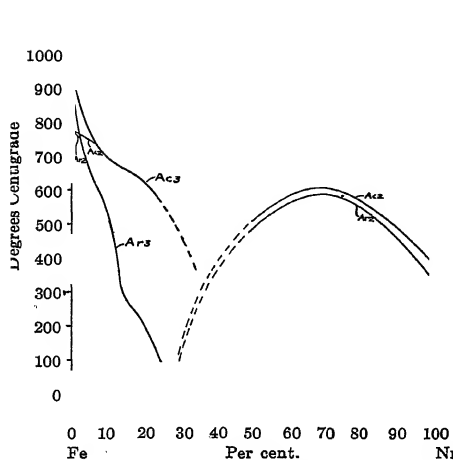


FIG. 1.—IRON-NICKEL STRUCTURE DIAGRAM.

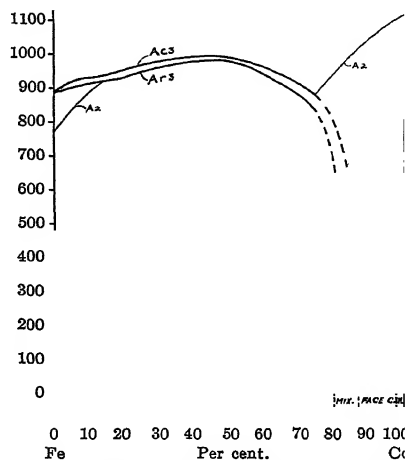


FIG. 2.—IRON-COBALT STRUCTURE DIAGRAM

of nickel a large temperature lag occurs between the critical points on heating and cooling, which in the case of cobalt is totally absent. The depression of the change point due to nickel is the chief cause of the nickel self-hardening steels, and the large temperature lag occasioned by the same element is responsible for the so-called irreversible steels. As the two characteristics causing these types are entirely absent in the iron-cobalt diagram their occurrence in the cobalt steels is naturally precluded.

The relationship of carbon with the alloying elements is a significant factor and has an important influence on the resulting steel. There is evidence that carbon forms the carbides Co_3C and Ni_3C , with cobalt and nickel, similar to Fe_3C , the carbide of iron. Both cobalt and nickel dissolve freely in iron, so it is not remarkable that they are found in steel occurring together with this element in the carbide. According to

Arnold,⁵ cobalt displaces the iron in the carbide to about 5 to 6 per cent. of its total proportion in the steel. Nickel⁶ was also found to displace the iron in much the same ratio.

The influence of nickel or cobalt on the eutectoid point of the iron-carbon alloys cannot be determined by an inspection of the binary diagrams given in Figs. 1 and 2. Scott⁷, Waterhouse⁸, and Kase⁹ have investigated ternary alloys containing iron, nickel and carbon and agree that the eutectoid point, in keeping with the allotropic change in the iron-nickel diagram, is lowered consistently by nickel additions. It was further noticed that the composition of the eutectoid was shifted to lower percentages of carbon with increased nickel percentages, and a large lag in temperature occurs in this point on heating and cooling. The A3 point is rapidly depressed by the effect of both nickel and carbon, and the lag occasioned by the influence of nickel on iron still persists in the presence of carbon.

EFFECT OF COBALT AND NICKEL IN TERNARY ALLOYS WITH IRON

In the ternary alloys of iron, cobalt, and carbon, the writer found the effect of cobalt on the critical points of steel to be markedly different from the effect of nickel. In some experiments carried out at the University of Sheffield the critical points of a number of plain carbon-cobalt steels were determined. The analyses of these steels are given in Table 1. The last five steels in the table were made available through the researches of Arnold and Read⁵ on the carbides of steel. The remaining alloys were prepared in a high-frequency induction furnace from Swedish pig and bar iron and a rundle cobalt of high purity. Thermal curves were taken by the inverse rate method and are reproduced in Fig. 3. The critical temperatures were determined at the first indication of a change on heating or cooling. (Table 2.) The exact temperature of the Ac3 point was rather difficult to determine and is given in the table as the temperature of the end of the change.

On comparison of the temperatures of the critical points on heating and cooling in Table 2, evidently no marked lag may be attributed to the influence of cobalt.

⁵ J. O. Arnold and A. A. Read: The Chemical and Mechanical Relations of Iron, Cobalt and Carbon. *Proc. Inst. Mech. Eng.* (1915, March), 247.

⁶ J. O. Arnold and A. A. Read: The Chemical and Mechanical Relations of Iron, Nickel and Carbon. *Proc. Inst. Mech. Eng.* (1914, March), 248.

⁷ H. Scott: Critical Ranges of Some Commercial Nickel Steels. *Trans.* (1922) 57, 100.

⁸ G. B. Waterhouse: The Influence of Nickel-carbon Iron. *Jnl. Iron Steel Inst.* (1905, No. II) 68, 376.

⁹ T. Kase: *Sci. Rept. Tohoku Imp. Univ.* No. 2, 14, 173.

In Fig. 4a and b, the Ar1 and Ar3 points of the cobalt steels have been plotted as functions of cobalt and carbon. Contours have been drawn at every 20° F. to represent the curvature of the temperature surfaces. From Fig. 4a, it may be observed that the A1 temperature is very slightly influenced by carbon additions, but that a consistent increase of about 8° F. is caused by each added per cent. of cobalt. In Fig. 4b, it is seen that the A3 temperature is affected by the addition of both elements, but the usual sharp decline due to carbon is somewhat mitigated by the influence of cobalt, which was found to raise the A3 point of pure iron.

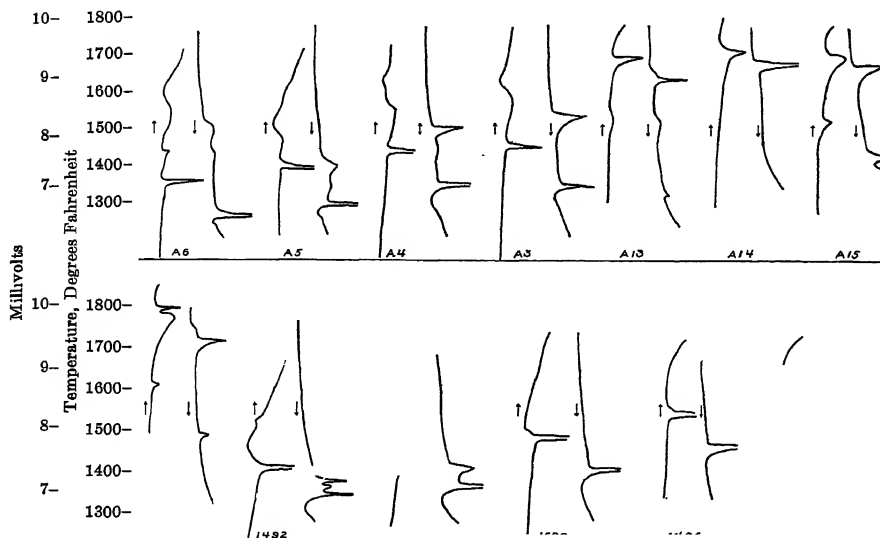


FIG. 3.—INVERSE RATE CURVES OF COBALT STEELS.

With such a few steels it is impossible to determine definitely the effect of cobalt on the eutectoid point. From the available information, cobalt appears to shift the composition in neither direction. In the two steels containing 0.84 and 11.18 per cent. CO, and 0.93 and 16.97 per cent. Co, the thermal curve exhibits only a single arrest point. The micrograph of the first of these steels (Fig. 5) displays a typical eutectoid structure. These facts would indicate that cobalt has very little effect on the eutectoid composition, even up to concentrations of 16 per cent.

The main conclusions that can be taken from the writer's investigation are that cobalt additions to steel elevate the A1 point; retard the decline, due to carbon, of the A3 point; cause no lag in the thermal points; and do not appear to affect the eutectoid composition.

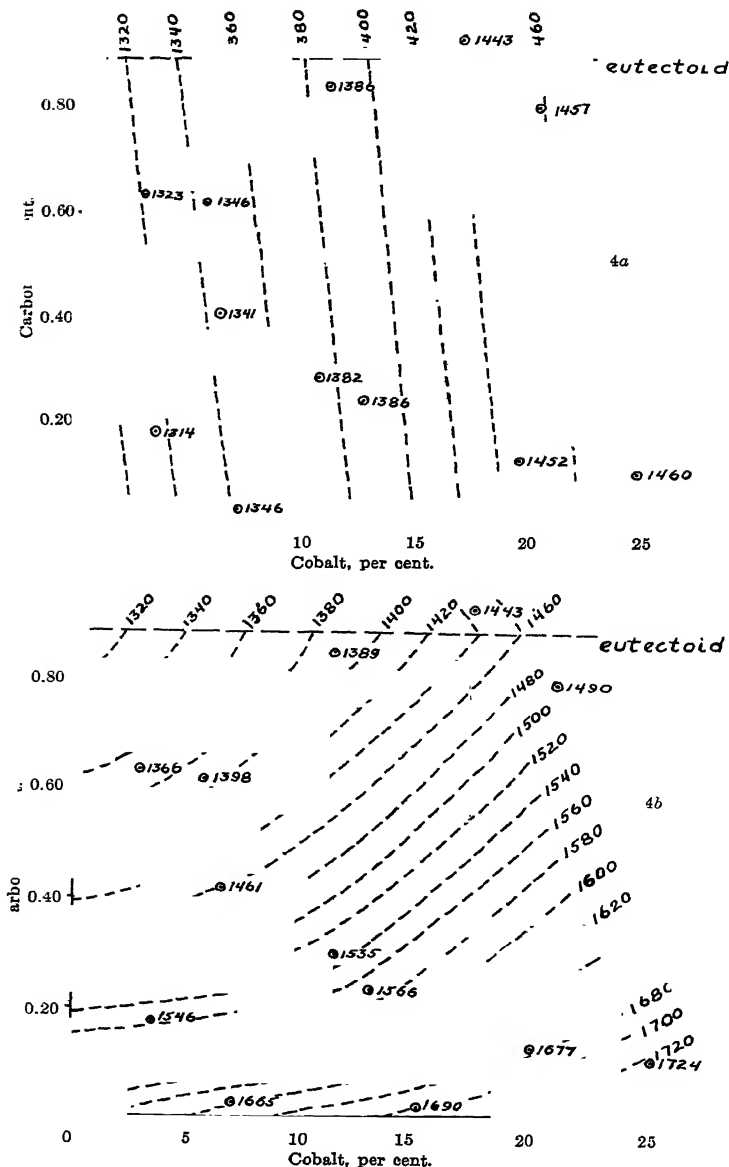


FIG. 4.—A1 (TOP) AND A3 (BOTTOM) POINTS ON COOLING OF COBALT STEELS IN DEGREES FAHRENHEIT.

TABLE 1.—*Critical Points in Plain Carbon-cobalt Steels*

Steel No.	Total Carbon, Per Cent.	Cobalt, Per Cent.	Manganese, Per Cent.
A3	0.24	12.89	0.19
A4	0.29	11.08	0.17
A5	0.41	6.24	0.16
A6	0.18	3.43	0.04
A7	0.38	5.83	0.03
A13	0.03	6.96	0.09
A14	0.02	15.07	0.06
A15	0.13	20.05	0.09
A16	0.10	25.18	0.10
A17	0.09	24.20	0.18
1492	0.64	2.68	0.16
1493	0.62	5.50	0.07
1500	0.84	11.18	0.23
1495	0.93	16.97	0.23
1496	0.79	20.85	0.25

TABLE 2.—*Thermal Critical Points of Iron-cobalt-carbon Alloys*

Steel No.	°F			°F			°F		
	Ac1	Ar1	Δ	Ac2	Ar2	Δ	Ac3	Ar3	Δ
A6	1389	1314	75	1467	1460	7	1620	1546	74
A5	1424	1341	83				1529	1461	68
A4	1467	1382	85				1586	1535	51
A3	1471	1386	85				1625	1566	59
A13		1346		1560	1553	7	1703	1665	38
A14							1722	1690	32
A15	1544	1452	92				1713	1677	36
A16	1580	1460	120				1768	1724	44
1492	1378	1323	55				1437	1366	71
1493	1407	1346	61				1472	1398	74
1500	1456	1386	70				1483	1389	94
1495	1511	1443	68				1555	1433	122
1496	1528	1457	71				1610	1490	120

PRECIPITATION OF CARBON IN COBALT AND NICKEL STEELS

A curious effect noticed in both cobalt and nickel steels on annealing is the decomposition of the carbides. Kase¹⁰ found that large quantities of carbon are precipitated in the solid state on cooling saturated iron-nickel alloys from the melt. The same author reports that nickel additions decrease the amount of combined carbon found in the cold alloys in almost a straight line ratio until not more than 0.40 per cent. com-

¹⁰ T. Kase: *Op. cit.*

bined carbon is found with 20 per cent. nickel. Waterhouse¹¹ mentions that on annealing well a 3.8 per cent. nickel steel containing 1.50 per cent. carbon more than 80 per cent. of the carbon is precipitated. Mathews¹² confirmed this and indicated that some of the carbon redissolves above the A1 point. Arnold and Read¹³ in their experiments with nickel steel, found the precipitation of carbon to be so vicious in the hammered bars that the metal had to be used as cast in order to obtain any amounts of the carbide.

In similar work on cobalt steels the same authors did not encounter this difficulty and found very little graphite in the hammered bars containing even as much as 15 per cent. cobalt. By annealing well, however, the carbon was almost entirely precipitated in the steels containing more than 2.5 per cent. cobalt. The writer experienced no difficulty due to graphitization in cobalt steels until concentrations of 15 per cent. cobalt were attained.

Fig. 5 shows the structure of an 11.18 per cent. cobalt steel containing 0.84 per cent. carbon, after having been heated to 1600° F. for 1 hr. and slowly cooled in the furnace overnight. The surface of this specimen was examined for fine graphite by a picric acid etch followed with a light polish, but no evidence of free carbon could be detected. The effect of the same heat treatment on high and low-carbon steels containing over 15 per cent. cobalt is shown in Figs. 9 and 10. In these steels the graphite has been precipitated in large quantities, and it is interesting to note that in the steel shown in Fig. 9, the carbide has been expelled to the ferrite grain boundaries, due to annealing. Before annealing this steel had the structure shown in Fig. 7.

The sparseness of the available data and the lack of uniformity of experiments make it difficult to determine whether cobalt or nickel has the more deleterious effect on the combined carbon. Arnold and Read consider that nickel carbide is the less stable because considerable carbon is precipitated in hammered bars containing only 7 per cent. nickel, whereas no noticeable precipitation occurs in the hammered bars con-

¹¹ G. B. Waterhouse: *Op. cit.*

¹² J. A. Mathews: Discussion of the Influence of Nickel and Carbon on Iron, by G. B. Waterhouse (Footnote 8). *Jnl. Iron Steel Inst.* (1905, No. II) 68, 402.

¹³ J. O. Arnold and A. A. Read: *Op. cit.*

FIG. 5.—COBALT, 11.18 PER CENT.; CARBON, 0.84 PER CENT.; ANNEALED 1 HR. AT 1600° F., FURNACE COOLED. PICRIC ACID ETCH. $\times 100$.

FIG. 6.—SAME STEEL AS FIG. 5. QUENCHED IN BRINE FROM 1560° F. PICRIC ACID ETCH. $\times 350$.

FIG. 7.—COBALT, 25.18 PER CENT.; CARBON, 0.10 PER CENT.; FORGED BAR. PICRIC ACID ETCH. $\times 100$.

FIG. 8.—SAME STEEL AS FIG. 7. QUENCHED IN BRINE FROM 2325° F. PICRIC ACID ETCH. $\times 350$.

FIG. 9.—SAME STEEL AS FIG. 7. ANNEALED 1 HR. AT 1600° F., FURNACE COOLED. GRAPHITE 0.06 PER CENT. PICRIC ACID ETCH. $\times 150$.

FIG. 10.—COBALT, 16.97 PER CENT.; CARBON, 0.93 PER CENT. ANNEALED 1 HR. AT 1600° F., FURNACE COOLED. PICRIC ACID ETCH. $\times 100$.

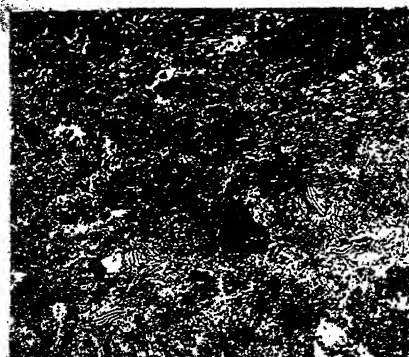


Fig. 5

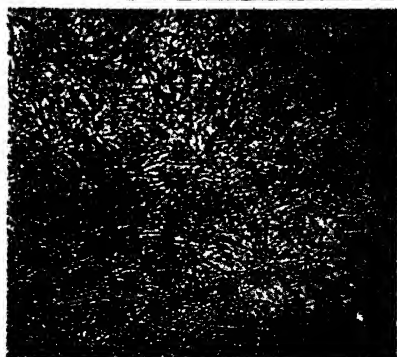


Fig. 6



Fig. 7



Fig. 8



Fig. 9



Fig. 10

taining as much as 16 per cent. cobalt. It is also true that on annealing well, the 3.82 per cent. nickel steel of Waterhouse precipitated as much carbon as the 5.5 per cent. cobalt steel of Arnold and Read; but the former contained over twice as much carbon as the latter. The evidence, although not altogether conclusive, appears to be that nickel causes precipitation more readily than cobalt. In any case, it is definitely certain that both elements will cause serious graphitization in plain carbon steel on prolonged annealing.

STRUCTURE OF COBALT AND NICKEL STEELS

The microstructure of the cobalt and the nickel steels bears out the characteristics of their respective ternary diagrams. The structure of the nickel steels has been investigated by many authors¹⁴ and it is generally accepted that, as the A3 temperature falls with increasing amounts of nickel and carbon, the steels that are pearlitic at low percentages become, with increasing additions, successively martensitic and finally austenitic even on slow cooling.

Cobalt steels, however, remain pearlitic over the entire range up to at least 40 per cent. cobalt, according to Guillet.¹⁵ Arnold and Read's micrographs show a pearlitic nature in a series of 0.80 per cent. carbon steels up to 20 per cent. cobalt. Fig. 7 shows the same structure occurring in a very low-carbon steel containing 24 per cent. cobalt. On quenching, the structure of cobalt steels becomes martensitic. A typical hardened structure is shown in Fig. 6. It was found necessary to quench the cobalt steels in brine from a temperature about 90° F. above the A3 point in order to render them martensitic. Hardening cracks frequently developed in steels containing about 0.75 per cent. carbon with low percentages of cobalt, but no cracks were found in steels containing more than 10 per cent. cobalt. It was found impossible materially to harden iron-cobalt alloys containing less than 0.15 per cent. carbon.

The structure resulting from quenching this type of alloy in brine from a temperature 560° F. above the A3 point is shown in Fig. 8. The Brinell hardness increased slightly from 168 to 238 due to quenching; but the steel in the harder condition shown in the micrograph possessed an intensity of magnetization of 1790, revealing that most of the structure consists of ferrite.

MECHANICAL PROPERTIES OF NICKEL AND COBALT STEELS

The mechanical properties are the criteria by which the usefulness of a given steel are commonly judged. Figs. 11 and 12 have been constructed from data concerning the mechanical properties of nickel and

¹⁴ Invar Nickel Steels. U. S. Bur. Standards *Circ.* 58.

¹⁵ Léon Guillet: Quarternary Steels. *Jnl. Iron Steel Inst.* (1906, No. II) 70, 1.

cobalt steels taken from the works of Guillet, Arnold and Read respectively. In these 0.80 per cent. carbon steels it is seen that the first few per cent. of nickel or cobalt have the effect of gradually raising the ultimate strength and the elastic limit, at the same time causing a slight decrease in the ductility. Beyond 3 per cent. nickel the powerful influence which this element exerts on the critical points begins to manifest itself. The microstructure of the nickel steels tends to become sorbitic or martensitic with a consequent abnormal increase in the

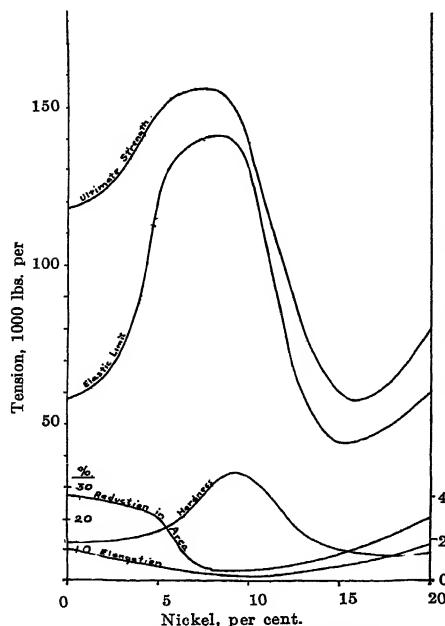


FIG. 11.—MECHANICAL PROPERTIES OF NICKEL STEEL, 0.80 PER CENT. CARBON (AFTER GUILLET).

(Brinell numbers on the right of each figure.)

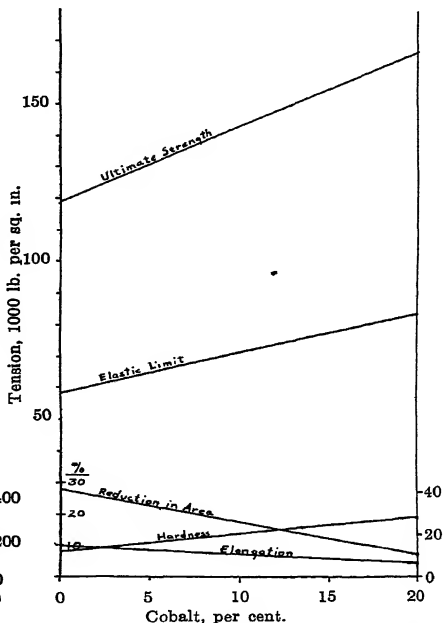


FIG. 12.—MECHANICAL PROPERTIES OF COBALT STEEL, 0.80 PER CENT. CARBON (AFTER ARNOLD).

ultimate strength, the elastic limit, and the elastic ratio. A corresponding decrease is also noticed in the ductility but it is relatively slight compared with the enormous increase in the tensile properties.

In the case of cobalt, however, there is no drastic effect on the critical points, and the mechanical properties continue to alter gradually with increasing cobalt in much the same proportion as is indicated by the addition of the first few percentages. The ductility becomes slightly reduced, the ultimate strength is increased, but the elastic ratio remains almost constant with increasing cobalt.

SUMMARY

From the foregoing considerations the effects of nickel and cobalt in steel may be summed up in the following statements. Chemically, both elements enter into the steel in much the same manner. They are found to dissolve freely in the ferrite, and are found also in solution with the iron in the carbide. Both render the carbide unstable, and nickel is probably the more deleterious agent.

The differences between the nickel and the cobalt steels are caused by the individual effects which these elements exert on the critical points. Nickel, by lowering the critical temperatures, benefits the steel with a grain refinement and a more uniform distribution of the carbide. Cobalt, on the other hand, raises the critical temperatures and thus permits a greater opportunity for the segregation of the microscopic constituents. In the nickel steels, this refinement of structure causes an improvement in the tensile and endurance properties which are not met with in the less uniformly constituted cobalt steels. To offset the slightly greater tendency for coarse structure occurring in cobalt steels, the alloying element has to offer only the gradual change in the mechanical properties which a dissolved metal generally confers on its solute.

ACKNOWLEDGMENTS

The writer wishes to acknowledge his indebtedness to Prof. Cecil H. Desch and the metallurgical staff at the University of Sheffield for their generous advice and assistance in the work on cobalt steels; and also his gratitude to Dr. J. A. Mathews for his kind suggestions in the preparation of this paper.

Optical Temperature Measurements in Open-hearth Furnace*

By B. M. LARSEN† AND J. W. CAMPBELL,‡ PITTSBURGH, PA.

(Pittsburgh Meeting, October, 1926)

SEVERAL articles have recently been published discussing the conditions necessary for accurate measurements of temperatures in the open-hearth steel furnace. In the course of a study of refractories service conditions in steel furnaces, the present writers have taken temperatures in several open-hearth shops in the Pittsburgh district. The conclusions, as to conditions necessary for accurate measurements of temperature, are somewhat at variance with those of other investigators.

The subject is closely connected both with the close control of steel-making and the prevention of overheating of the silica refractories used in roofs and walls. It is becoming more and more important to obtain furnace temperatures of fairly definitely known accuracy. At present, the most useful instrument for such measurements is the disappearing-filament type of optical pyrometer. This method of measurement has formed the basis of most other discussions on the subject and was also used by the writers.

PREVIOUS INVESTIGATIONS

In 1916, Burgess¹ made perhaps the first systematic series of measurements in steel furnaces. The emissivity figures of 0.40 for liquid steel and of 0.53 to 0.65 for liquid slags had been determined by the Bureau of Standards and were used by Burgess in this work. These correction factors are still being used and are regarded as fairly accurate, so that the tapping and teeming temperatures on metal streams given by Burgess should be very nearly correct. Most of his temperatures of furnace roofs and slag surfaces were taken while flames were present in the furnaces and include a few roof temperatures above 1700° C. (3100° F.).

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† Assistant Metallurgist, U. S. Bureau of Mines, Pittsburgh Experiment Station.

‡ Research Fellow, 1924-25, Carnegie Institute of Technology.

¹ G. K. Burgess: Temperature Measurements in Bessemer and Open-hearth Practice. Bureau of Standards *Tech. Paper* No. 91 (1916).

In 1918, Johns² gave an account of some temperature measurements in acid open hearths. He included a table in which melting chamber temperatures varying from 1700 to 1800° C. (3100 to 3275° F.) are given as "corrected" temperatures. Johns assumed that the walls in the chamber approached closer to the condition of free radiating surfaces than that of black body radiation. He therefore corrected the observed temperatures upward by assuming an emissivity factor of 0.56 for the brick surfaces and of 0.50 for slag. His uncorrected temperatures (taken just after tapping with flame out of the furnace) vary from 1575 to 1665° C. (2870 to 3030° F.). These agree very well with those obtained by the writers.

In 1922-23, Greenwood³ made an extended study of the errors and corrections involved in optical temperature measurements in steel-making. He recognized the fact that reflection from flames increased the observed temperature, and he studied the differences between "flame-on" and "flame-off" temperatures, the latter being taken through an open door during reversals. He concluded that the amount of increase in apparent temperatures by reflected light from flame is fairly constant at around 27 to 35° C. (49-63° F.). Calculated temperatures of the slag surface at time of tapping were obtained, based on the temperature of tapping streams of slag and of metal. These calculated values were found to be uniformly near to 35° C. (63° F.) higher than corresponding temperatures taken on the slag surface through an open door just before tapping with flame out of the furnace chamber. The natural conclusion was that the chamber with flame absent is not a true black body, but that the lowering error caused by this condition is just about balanced by the high error caused by flame reflection. If these results correctly portray conditions, then observations through an open door with flame in the chamber should give very nearly true temperatures of slag and wall surfaces. Greenwood then concludes that if a small peephole were used to sight through instead of a large door wide open, flame-off readings would give true temperatures.

POSSIBLE ERRORS IN OPTICAL MEASUREMENTS

The first question is whether or not the open-hearth melting chamber *with flame shut off* is at or very near the condition of a black body or perfect radiator, during most of the operation of steel-making. Measurements made by the writers have indicated that this condition holds at

² Cosmo Johns: The Determination of the Temperature of Liquid Steel Under Industrial Conditions. *Trans. Faraday Soc.* (1918), **13**, Part 3, 280.

³ J. N. Greenwood: Determination of the Corrections to Optical Pyrometer Readings Taken During Steel-making and Casting. *Carnegie Schol. Mem., Iron Steel Inst.* (1923), **12**, 27.

all periods excepting probably during the charging and early melting periods. This conclusion is in direct contradiction to that of Johns and somewhat different from those of Greenwood.

If we find that flame-off temperatures, taken through a small opening leading into the melting chamber, are true temperatures, then by comparison it is possible to determine the effect of flames, wide-open doors, etc., on the accuracy of these measurements. The experimental steps are then as follows: (1) Determination of the accuracy of flame-off optical temperatures on wall and roof surfaces by comparison with some other method of measurement; (2) measurements of flame-on and flame-off temperatures to determine the effect of flame reflection under different conditions; and (3) measurements of temperatures on roof, wall and slag surfaces, etc., to determine the distribution of temperatures in different parts of the melting chamber during steel-making.

TEMPERATURE GRADIENTS IN THE WALLS AND ROOF OF THE MELTING CHAMBER

Thermocouples were placed in several open-hearth roofs and walls and in each case measurements of temperature gradients were obtained

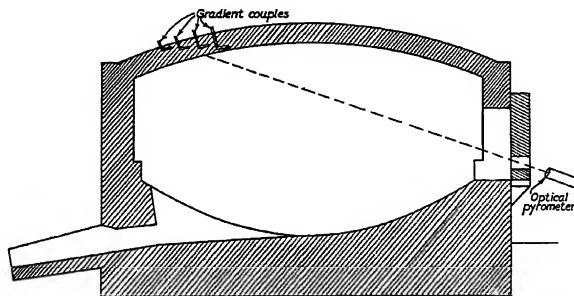


FIG. 1.—DIAGRAM SHOWING POSITION OF GRADIENT COUPLES IN ROOF ARCH AND LINE OF SIGHT FOR OPTICAL MEASUREMENTS ON ROOF SURFACE.

from these throughout several days of actual steel production. In each test, one platinum, platinum-rhodium couple was placed only a short distance (0.3 to 0.6 in.) from the inner surface of the arch or wall. Extrapolation of such gradient curves to the wall surfaces should give a very accurate measure of the inner surface temperatures, provided the couples accurately measure the temperatures at various planes through the brick. In the comparison of these extrapolated temperatures with optical pyrometer temperatures taken on the inner surfaces, we have a method for determining the accuracy of optical measurements. This is illustrated by Fig. 1. Gradient couples are placed in the brick of the roof arch in area A. During the test, optical measurements are made by

sighting on the inner roof surface of area A through a small opening in a furnace door, measurements being taken with flames both on and off in the melting chamber.

The gradient couple, circuits, etc., were carefully standardized. To obtain accurate gradient temperatures, it was then only necessary to be sure that the hot junctions of the thermocouples were actually at the temperatures of the brick substance along the isothermal planes in which they were placed. The only source of error involved here is that caused by the rapid conduction of heat away from the junction along the metal wires of the couple. This effect, which causes a couple to read low, may be prevented by placing a few inches of the wires at the hot end of the couple in the same thermal plane as the hot junction. Gradient couples are often placed like couple B in Fig. 2. Here the temperature of the isothermal plane AA' is to be measured. In couple B only the hot

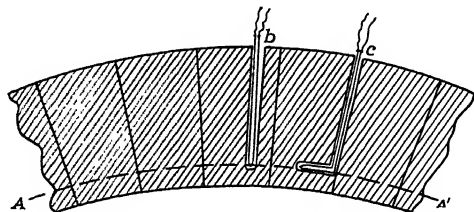


FIG. 2.—TWO METHODS OF PLACING GRADIENT COUPLES IN BRICK OF AN ARCH OR WALL.

junction is in this plane, and a rapid drop in temperature occurs along the wires above this junction. Under such conditions, the metal wires conduct heat away from the junction to cooler planes of the brick; this occurs rapidly enough so that the junction is below the true temperature of plane AA' . Couple C is placed so that the wires back of the junction are also in isothermal plane AA' ; heat is not drained away from the junction and it will register the true temperature of plane AA' . In a laboratory test the magnitude of this error for 24 B. and S. gage Pt.-Pt.-Rh. couples was found to be about constant at 40°F. (22°C.) at temperatures above about 600°F. (333°C.).

Fig. 3 gives some representative log temperature curves in this case for various depths 0.8, 4.5 and 6.9 in. respectively, in the 9-in. roof arch of a 50-ton basic open-hearth furnace. These curves cover the period of burning-in the furnace lining and the first five heats of steel produced. Couple No. 1 was placed 0.7 in. from the inner roof surface. (In this test another platinum couple was placed only about $\frac{1}{8}$ in. back from the inner surface; it failed very soon after the preheating period.) These curves show the wide variations in temperature which occur in the inner zone of the walls and arch between heats and during periods of scrap charging and bottom repairs. The outer zone of the arch layer is comparatively constant in temperature during steel-making, from about 500 to 600°F. (275 – 330°C.).

Extrapolation of the gradient curves obtained from Fig. 3 indicated that the inner surface was usually about 100 to 120° F. (55-65° C.) above the temperatures given by couple No. 1.

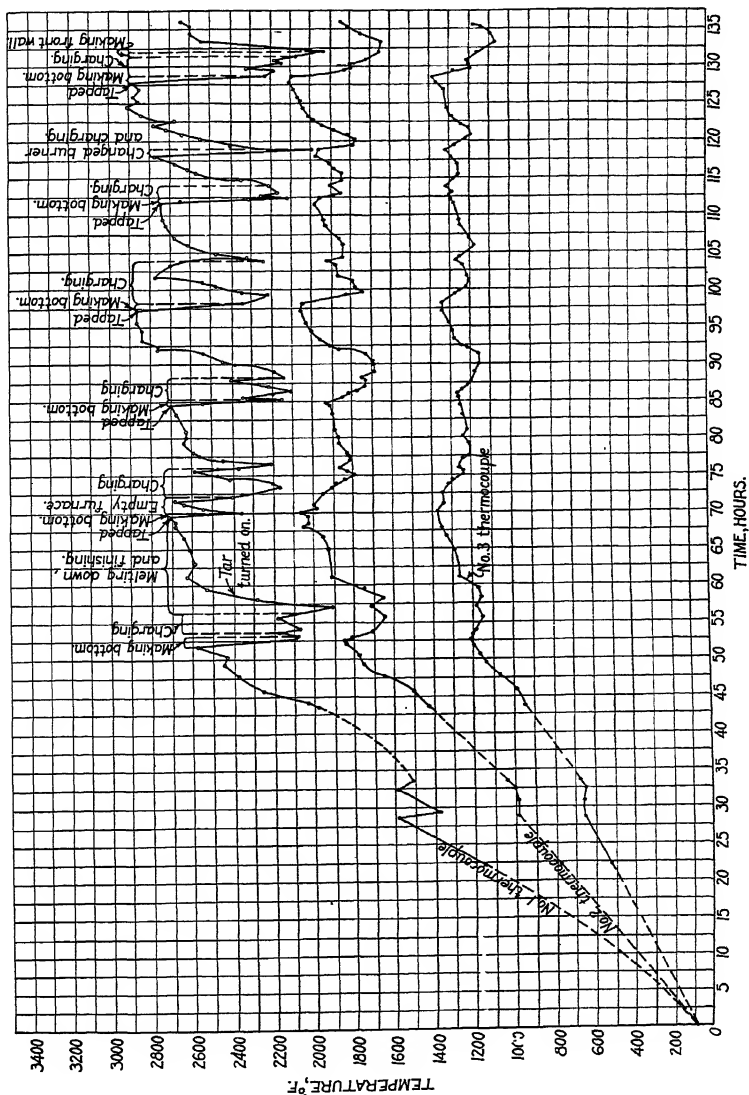


Fig. 2. Total suspended matter, mg/m³ in the surface layer of the water.

From these data we find that during periods of easy working, maximum melting chamber temperatures were in the range of 2700 to 2830° F.

(1480–1555° C.). During periods in which the furnace was pushed to the limit and slight fusion of silica brick occurred, these maximum temperatures were about 2950 to 3000° F. (1620–1650° C.). Other data have shown that, in general, open hearths with silica roofs are limited to the rather narrow working range of about 2700 to 3000° F. (1482–1650° C.). The lower limit is that fixed by the melting point of steel and the upper one by the softening point of flux-saturated silica arch brick.

During this test, gradients in the back wall were also determined. Optical temperatures were taken on inner back wall and roof arch surfaces on the areas in which the gradient couples were placed, as illustrated

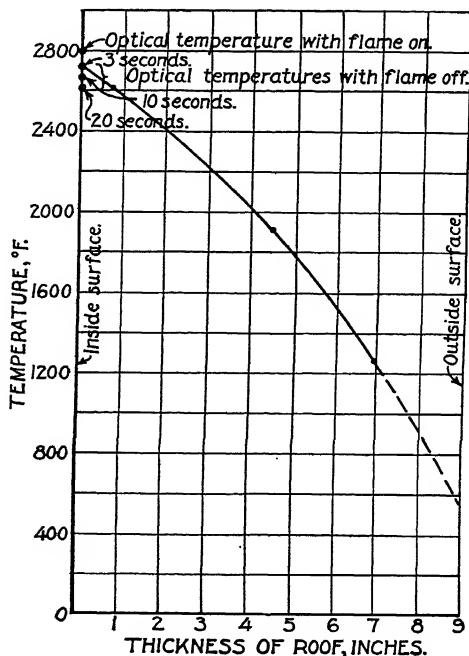


FIG. 4.—TEMPERATURE GRADIENT FOR ROOF OF A BASIC FURNACE.

in Fig. 1. Table 1 gives a comparison between a number of these optical temperatures and the surface temperatures indicated by the gradient curves. The latter were obtained by first drawing a smooth gradient curve to the inner roof surface and placing on the graph the corresponding optical temperature, to eliminate the personal factor. Fig. 4 shows one such curve for the roof and Fig. 5 two curves for the back wall.

In Table 1, 11 out of 13 flame-off optical temperatures check with the extrapolated temperatures to within 0 to 10° F. (0 to 5° C.) and in the other two cases the differences are 35 and 43° F. (20 to 25° C.). These checks are obtained throughout the temperature range of 1900 to 2900° F. The extrapolation of gradient curves through 0.7 in. of brick introduces a possible uncertainty, but the evidence on the whole at least strongly favors the conclusion that flame-off optical temperatures are accurate. From Table 1 we also find that the flame-on optical temperatures are always appreciably higher than the extrapolated temper-

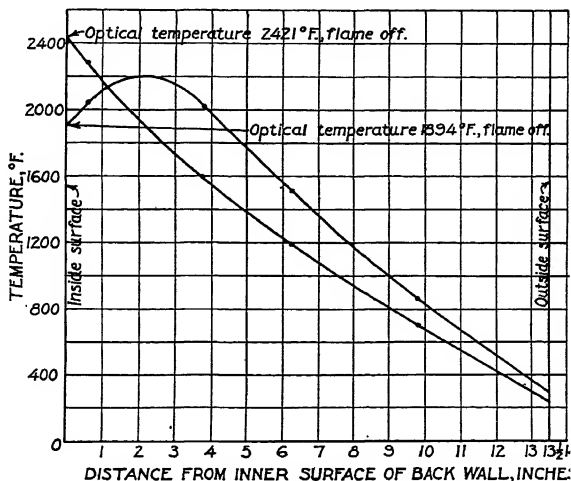


FIG. 5.-TEMPERATURE GRADIENTS IN CHROME BACK WALL OF A 50-TON BASIC FURNACE.

atures, these differences ranging from 35 to 240° F. (20 to 135° C.). These results were checked almost exactly by another test in a larger furnace.

In taking flame-off optical temperatures on melting chamber surfaces, it must be remembered that these surfaces cool very rapidly after the flame is cut off from the furnace. This effect is shown in Fig. 4. The temperature gradient for the roof is given for a period in which the furnace was working slow and was running along with rather steady conditions. The indicated surface temperature for the roof is about 2715° F. (1490° C.), about 75° high. The flame was then shut off, and an optical reading taken as quickly as possible within 3 sec. gave 2710° F. (1488° C.). The roof was then cooling rapidly; in 10 sec. it was down to 2660° F. (1460° C.) and in 20 sec. to 2610° F. (1432° C.), an approximate

TABLE 1.—*Comparison of Optical Temperatures with Temperatures Obtained by Extrapolation of Gradient Curves*

Time after Starting Furnace		Condition of Flame	Place in Furnace	No. 1 Thermo- couple		Extrapolated Surface		Optical		Difference	
Hr.	Min.			Deg. F.	Deg. C.	Deg. F.	Deg. C.	Deg. F.	Deg. C.	Deg. F.	Deg. C.
52	40	On On	Roof Back Wall	2187 2158	1197 1181	2240 2280	1226 1249	2484 2428	1362 1331	244 148	136 82
58	10	Off Off On	Back Wall Roof Roof	2283 2435 2290	1250 1335 1254	2430 2400	1332 1316	2421 2435 2547	1327 1335 1397	9 35 147	5 20 82
63	40	On Off	Roof Roof	2610	1452	2715	1490	2797 2712	1536 1489	82 3	46 1
84	40	Off On	Roof Roof	2705	1485	2820	1549	2828 2900	1353 1593	8 80	4 44
85	10	Off Off	Roof Back Wall	2185 2045	1196 1118	1910 1900	1043 1038	1905 1894	1041 1034	5 6	3 4
107	45	On	Roof	2705	1485	2820	1549	2880	1582	60	33
108	45	Off On	Roof Roof	2725	1506	2840	1560	2835 2930	1557 1610	5 90	3 50
109	50	On	Roof	2733	1511	2850	1566	2910	1599	60	33
111	10	Off On	Roof Roof	2733	1510	2835	1557	2840 2910	1560 1599	5 75	3 42
133	30	On	Roof	2535	1390	2715	1490	2750	1510	35	20
134	30	Off	Roof	2595	1424	2710	1488	2713	1489	3	1
135	15	Off On	Roof Roof	2595	1424	2715	1490	2720 2840	1493 1560	5 135	3 75
136	0	Off On	Roof Roof	2640	1449	2760	1516	2750 2860	1510 1571	10 100	5 55
166	35	Off On	Roof Roof	2773	1523	2865	1574	2822 2870	1550 1522	43 5	24 3
167	5	Off On	Roof Roof	2773	1523	2870	1522	2870 2930	1522 1610	0 60	0 33

cooling rate of 5° F. (2.8° C.) per sec. Readings taken immediately after shutting off the flame seem to give nearly true operating furnace temperatures.

DIFFERENCES BETWEEN FLAME-OFF AND FLAME-ON OPTICAL READINGS

Table 2 includes a number of pairs of optical temperatures taken on roofs with flame on and off in the melting chamber. In these measure-

ments, the pyrometer was first sighted on the roof surface above the flame zone—that is, with no flame in the field of vision—to obtain “flame-on” temperatures. The flame was then shut off and another temperature taken immediately, on the same area of the roof.

TABLE 2.—*Optical Temperatures on Roof Surfaces with Flame Off and On in the Furnace*

Type of Furnace	Fuel Used	Roof Surface Temperatures				Difference	
		Flame Off		Flame On			
		Deg. F.	Deg. C.	Deg. F.	Deg. C.	Deg. F.	Deg. C.
45-ton basic.....	Natural and by-product gas—rich mixture	2950	1621	2980	1638	30	17
		3010	1656	3040	1671	30	15
		2985	1640	3050	1677	65	37
		2895	1590	2950	1621	55	31
		2930	1610	3010	1656	80	46
45-ton basic.....	Natural and by-product gas—lean mixture	2920	1604	3010	1656	90	52
		2900	1593	2920	1604	20	11
		2955	1624	2995	1646	40	22
		2935	1613	2960	1627	25	14
		2930	1610	2960	1627	30	17
80-ton basic.....	Tar	2870	1577	3050	1677	180	100
		2870	1577	2980	1638	110	61
		2780	1527	2905	1596	125	69
		2750	1510	2870	1577	120	67
50-ton basic.....	Tar	2780	1527	2880	1582	100	55
		2740	1505	2905	1596	165	91
		2880	1582	3000	1649	120	67
300-ton Talbot.....	Tar and by-product gas	2830	1554	2930	1610	100	56
350-ton Talbot.....	Producer gas	2865	1574	2930	1610	65	36
		2845	1563	2880	1582	35	19
		2850	1566	2900	1593	50	27
		2830	1552	2940	1615	60	33
		2870	1577	2960	1626	90	49
20-ton acid,	Oil	2900	1593	2970	1632	70	39
		2910	1599	2990	1643	80	44
		2920	1604	2985	1640	65	36
		2970	1632	3060	1683	90	51
		2290	1254	2505	1374	215	120
		2325	1274	2610	1432	285	158

The presence of flame always has the effect of raising the apparent roof or wall temperature due to reflection by the wall of more intense radiant energy from the flame. Contrary to the conclusions of Greenwood,⁴ however, the amount of this effect seems to vary greatly in different furnaces and at different times in the same furnace. The differences given by the table vary from 20 to 285° F. (11 to 160° C.) Many factors affect this error. Brighter flames from tar and oil have a larger

⁴ J. N. Greenwood: *Op. cit.*

effect than relatively non-luminous flames such as those from cleaned by-product coke-oven gas. Raising the apparent flame temperature and lowering the true wall temperature both increase the effect of flame reflection. Glazed surfaces reflect more radiant energy than rough surfaces. As an extreme case of this sort, a quiet slag with a mirror-like surface would reflect nearly all the light from the flame and give an apparent temperature very near to that of the flame itself.

Simultaneous readings with flame off on different areas of a roof have indicated that the roof arch tends to be very nearly uniform in temperature over its whole surface. Readings taken with flame on, however, give apparent temperatures which vary considerably from end to end of the roof surface. At the end just above the flame (at its greatest width and brightness) the apparent temperature is highest because the roof arch here reflects more of the flame radiation. Such apparent temperatures may be from 35 to 100° F. (20 to 55° C.) higher at the incoming flame end of the roof. The true roof temperatures are usually the same at both ends, and lower than any of the apparent flame-on temperatures.

The tendency mentioned for true temperatures to approach uniformity over the whole roof area does not always result in such conditions in practice. What we mean is that through the rapid interchange of heat by radiation between various surfaces in the melting chamber at such high temperatures, there is a strong force tending to equalize these temperatures. Frequently, however, the flow of heat to one area will be excessively rapid, due to a current of hot gases impinging there or because it is close to the flame and this surface will be somewhat hotter than other parts of the roof.

TEMPERATURE DISTRIBUTION IN THE MELTING CHAMBER

The distribution of temperatures between roof, walls and bath in the melting chamber is important, because it helps to explain the failure of refractories in roof and walls and also the mechanism of heat transfer in the furnace. Conditions are variable enough in different furnaces so that the only feasible method of determining general conditions is to coordinate a large number of observations and measurements in different shops.

In a furnace near the end of a heat, with a fairly quiet slag covering on the bath and the flame shut off, very little difference in apparent temperatures can be noted with the eye alone. Also, the close agreement between optical and thermocouple measurements with flame out of the chamber shown in Table 1 would suggest that the chamber as a whole must be close to "black body" conditions—that is, uniform in temperature throughout.

Table 3 includes a number of comparative temperatures of roof and back walls taken in several widely different types of furnaces. For each pair of temperatures, both measurements were taken simultaneously on roof and back wall with two optical pyrometers and with no flame in the furnace. Temperature differences obtained vary between 0 and 60° F. (0 to 33° C.), with the back wall sometimes hotter and sometimes cooler than the roof. The average difference is only 14° F. (8° C.), while the probable error in measurements is about 5 to 10° F., indicating a definite tendency here toward uniformity of temperatures.

As the furnace process is essentially one of supplying heat to steel and slag to form the liquid bath, one should expect to find a tendency toward lower temperatures in the slag as compared to those on roof and wall surfaces. This tendency is shown by the comparative temperatures of slag and wall surface given in Table 4. These measurements were

TABLE 3.—*Comparative Temperatures of Roof and Back Wall Surfaces of Five Types of Open-hearth Furnaces*

Type of Furnace	Roof		Back Wall		Temperature Difference	
	Deg. F.	Deg. C.	Deg. F.	Deg. C.	Deg. F.	Deg. C.
50-ton basic, stationary	2845	1563	2846	1563	1	0
	2805	1540	2850	1566	45	26
	2770	1521	2784	1529	14	8
	2785	1529	2778	1525	7	4
	3010	1656	2950	1621	60	35
	2895	1590	2840	1560	45	25
	2900	1593	2875	1579	25	14
45-ton basic, stationary	2920	1604	2970	1632	50	28
	2955	1624	2955	1624	0	0
	2935	1613	2940	1615	5	2
	2905	1596	2905	1596	0	0
	2860	1571	2860	1571	0	0
	2900	1593	2895	1590	5	3
	2955	1624	2955	1624	0	0
80-ton basic, stationary	2870	1577	2900	1593	30	16
	2860	1571	2870	1577	10	6
	2850	1566	2858	1570	8	4
350-ton Talbot, tilting	2852	1567	2846	1563	6	3
	2846	1563	2852	1567	6	3
	2820	1548	2815	1546	5	2
	2970	1632	2970	1632	0	0
	2910	1599	2912	1600	2	1
20-ton acid stationary	2820	1548	2834	1557	14	9
	2930	1610	2915	1602	15	8
	2930	1610	2941	1616	11	6
	2968	1631	2970	1632	2	1

TABLE 4.—*Comparative Temperatures of Slag and Wall Surfaces of Three Types of Open-hearth Furnaces*

Type	Roof		Back Wall		Slag		Maximum Difference	
	Deg. F.	Deg. C.	Deg. F.	Deg. C.	Deg. F.	Deg. C.	Deg. F.	Deg. C.
350-ton Talbot basic, tilting.	2846	1563	2845	1563	2745	1507	100	56
	2852	1567			2810	1543	36	20
	2846	1563	2852	1567	2835	1557	17	10
	2876	1580			2820	1548	32	19
50-ton basic, stationary.....	2735	1501			2810	1543	66	37
	2846	1563			2745	1507	10	6
	2875	1579			2780	1527	66	36
	2920	1604			2845	1563	30	16
20-ton acid, stationary.....	2906	1597			2870	1577	50	27
	2900	1593			2850	1566	56	31
	2910	1599			2834	1556	66	37
	2870	1577			2810	1543	100	56
	2870	1577			2840	1560	30	17
	2870	1577			2895	1590	25	13
	2905	1596			2895	1590	10	6
	2910	1599	2912	1600	2919	1603	9	4
	2924	1607			2910	1599	14	8
	2820	1548	2834	1556	2810	1543	24	13
	2864	1573			2822	1550	42	23
			2895	1590	2850	1566	45	24
	2930	1610	2915	1602	2870	1577	60	33
	2930	1610			2957	1625	27	15

taken in the same manner as those of Table 3. The slag surface is cooler than the roof of back wall in 18 out of 22 cases, the differences varying from 10 to 100° F. (5 to 55° C.), with an average of about 50° F. (28° C.). In a few cases near the end of a heat the slag was found to be hotter than the wall surfaces.

Slag surface temperatures are a less definite sort of value than readings made on wall surfaces because of rapid temperature variations due to convection currents in the liquid slag layer over the cooler steel below. The bubbles coming up in the slag during boils usually appear darker than the rest of the surfaces. Slag conducts heat comparatively slowly and a perfectly quiet slag layer would have surface temperatures, equal to, or higher than those in the wall surfaces. Most of the heat flow through the slag layer probably occurs by convection; the rapid stirring in the slag layer keeps the surface cooled somewhat below the rest of the lining surfaces, as it carries heat received by radiation from walls and flame to the steel below. Near the end of a heat the steel is heated close to the furnace temperature so that it can no longer cool the slag surface and

the latter may then become hotter than the furnace walls. When this occurs there is usually danger of overheating the roof surface unless the furnace is closely regulated.

After the steel is tapped from the furnace, the flame is shut off and the doors opened for patching bottom and banks. Hearth, wall and roof surfaces immediately begin to cool rapidly. Fig. 6 gives the cooling curve of the melting chamber surface in a 20-ton acid furnace during such a cooling period between heats. Temperature measurements

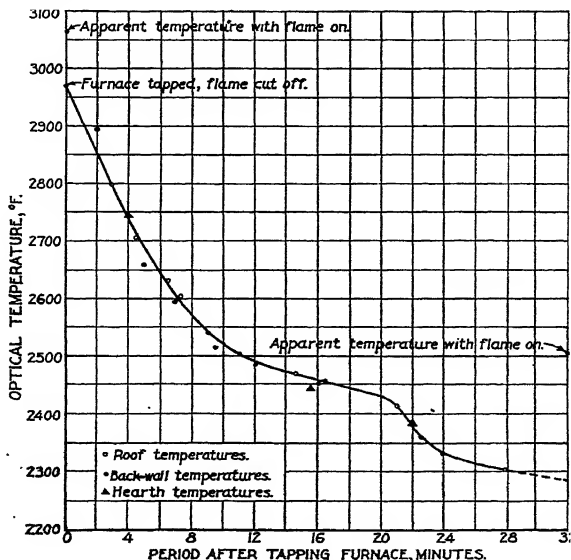


FIG. 6.—COOLING CURVE OF MELTING CHAMBER OF AN ACID OPEN-HEARTH FURNACE DURING PERIOD BETWEEN HEATS.

for this curve were taken with an optical pyrometer through an opening in one port end, sighting on areas in roof, back wall and hearth through the whole cooling period until the flame was turned on again to start charging in the next heat. The curve is drawn through the points representing roof temperatures, but all the back wall and hearth temperatures fall on or very near to this curve. Apparently the whole chamber cools down evenly, probably because the temperature is always kept nearly uniform by rapid radiation interchange of heat between the hearth, wall and roof surfaces. The lag in the middle of this curve was caused by shutting the furnace doors for a while, which slowed up the rate of heat flow out of the chamber.

Roof surface temperatures with "flame on" were taken just before the flame was shut off at tapping, and just after it was turned on again to

start the next heat. These points are shown in Fig. 6 above the ends of the cooling curve and show quite clearly the effect of flame reflection in raising the apparent wall surface temperatures. The effect is greater—over 200° F. (111° C.) at the lower temperature range, due to the wide difference between the true temperatures of flame and roof brick compared to the corresponding difference just before tapping.

CONCLUSIONS

1. At the high temperatures of steel making, the rapid interchange of heat tends to keep the various surfaces of roof, walls, bath, etc., in the open hearth at very nearly the same temperatures. Near the end of a heat, especially, temperatures are practically uniform throughout the melting chamber. During the charging and early melting periods the hearth and steel are absorbing heat rapidly, and their surfaces are probably much cooler than those of the side walls, ports and roof. By the time the slag covering has formed on the bath, however, this difference is in most cases apparently less than 100° F. (56° C.). As the bath, approaching the end of the heat, comes up in temperature, these differences grow still smaller.

2. The variations in heat flow through different parts of the roof, side walls, ports, etc., are apparently small, in comparison to the speed of radiation interchange between their surfaces at steel-making temperatures. Observations indicate that such surfaces are all at practically the same temperatures during steel making.

3. During the period of hearth repair between heats, the melting chamber surfaces cool rapidly, but the uniformity of temperature which exists just before tapping is still maintained. This, again, can only be attributed to the rapid interchange of heat by radiation.

4. The furnace chamber, then, approaches "black body" conditions so closely during most periods of operation, that flame-off optical temperatures are very nearly accurate. Near the end of a heat, when furnace temperatures are most needed, the furnace chamber becomes almost an absolute "black body."

5. Operating temperatures on wall surfaces, etc., must be obtained by reading the pyrometer immediately after the flame has been shut off, because of rapid cooling rates at such temperatures.

6. Flame reflection always raises the apparent temperatures of wall or slag surfaces. The amount of this effect, however, is extremely variable and the error cannot usually be accurately corrected by any factor. Even at the same time in different areas in the same furnace chamber, variable flame reflection sometimes indicates apparent differences in temperatures which are not actually present.

7. The range of steel-making temperatures with silica arches or walls is about 2700 to 3000° F. (1482 to 1649° C.). Temperatures just above

this range will cause the flux-saturated silica brick to slowly fuse and drip. Slow-working furnaces may have finishing temperatures down in the range of 2725 to 2800° F. (1496 to 1538° C.): fast-working furnaces usually finish a heat at around 2925 to 2975° F. (1607 to 1635° C.).

8. One area in a roof or wall may be overheated by impinging of a stream of hot gases or flame. If this does not occur, the "evening up" of temperatures between the roof and the heat-absorbing bath usually prevents overheating of the brick. With such conditions, the roof or walls are in danger of fusion only near the end of a heat—when both the walls and the bath come up together into the "danger zone" of temperatures above 2950° F. (1620° C.).

with the growth of world demand for minerals, and there is nothing on the horizon to stop this acceleration for a long time to come. The outlying countries possessing manganese have felt and will continue to feel this pressure, and must yield to it whether they like it or not. It is inevitable and inexorable. From our point of view fair methods are on the whole being used, and the exploitation results not only to our advantage but to the advantage of countries being exploited; but whether this is true or not, the exploitation will continue. In a way, this is the doctrine that "might makes right," but only to the extent that might is permanently lodged by nature in certain places. We are merely conforming to our environment. Our action in this field is no more right or wrong than is nature's inequality of environment.

As natural and reasonable as this course appears from our point of view, I think I am safe in saying that much of the public attitude toward exploitation is quite different. The word "exploit" originally meant to develop or get the value out of, but because of occasional abuses it has come to mean in the public mind "to take unfair advantage of," by more or less ruthless methods. There is a strong tendency to deplore all exploitation, or to assume that only other nations are doing it, or that if our nationals are doing it, they ought to stop. Exploitation of manganese is not discriminated from any other kind of commercial activity in foreign countries, essential or non-essential. It is called "commercial buccaneering," "economic imperialism," "dollar diplomacy," unwarranted interference with national self-determination, something that some bright persons thought of as a mere money-making venture, that might be stopped with loss only to those engaged, with no realization of its essential and inevitable character resulting from nature's distribution of the elements of our environment. Even some of the people who take it for granted that manganese must be secured will at the same time oppose "economic imperialism" or "exploitation" or interference with other nations without realizing the contradictions in their attitude.

When one stops to think of it, this confusion is not surprising. Governments have seldom been frank in explaining foreign exploitation. When they find it necessary to use pressure, political or military, it is almost uniformly announced as "protection of national interests," without explaining why or how these interests happen to be there. They have seldom discriminated between interests essential for national welfare and the less essential ones. Almost never have they clearly and frankly stated that exploitation was an inevitable consequence of the way in which material civilization has adapted itself to resources. The recent public statements of our government in regard to Mexico and Nicaragua have, so far as the general public is concerned, failed to bring home the fact that we are in fact exploiting these countries and must necessarily continue to do so.

NEED OF PROPER PERSPECTIVE ON MINERAL EXPLOITATION

What is needed is, first, a clear understanding that nature's distribution of sources requires international exploitation of certain minerals, if our material civilization is to go forward along present lines; that whatever theoretical considerations of right or wrong are involved, the movement cannot be stopped; that this movement requires the open door; that in some measure it requires the overriding of national self-determination; that it is in fact a phase of economic imperialism, however objectionable this term is; and that dollar diplomacy is involved. The hypocritical attitude should be dropped that we are not and should not be doing these things.

Until the political atmosphere is cleared on this question, there is always danger of snap political action, based on temporary and local circumstances, which may go to the one extreme of forcing exploitation on other nations without regard to their legitimate rights, or to the other extreme of handicapping necessary foreign exploitation by lack of national support.

Second, there is immediate and great opportunity for constructive effort directed toward the *manner* in which the job is being done, including a better understanding and definition of the open-door policy and of the rights of both the exploiter and the exploited. Experience has shown both good and bad ways of exploitation. As the leading exploiter of the world's minerals, it would seem reasonable to expect that sooner or later our country might formulate the principles of a policy which puts mineral exploitation in proper perspective, which discriminates necessary from unnecessary exploitation, and which could be adhered to with a little more consistency than has been exhibited in recent decades. Neither in the intelligent formulation of political principles of exploitation nor in the revision of methods of exploitation has the manganese industry anything to fear. It is necessary, and inevitable, and for the most part is now being done with due regard to the rights of other nations.

The public has not been adequately informed about the facts of manganese or even as to our country's vital need for it. Among users and technical men connected with the industry one finds occasionally only partial information or a sadly distorted perspective. There is room for much self-education, and particularly for careful thinking about the political problems involved. When an engineer is asked his views, on some of the political questions involved, the chances are better than even that he will be found not to have thought them through. It is this situation which has led the Mining and Metallurgical Society of America and the American Institute of Mining and Metallurgical Engineers to appoint committees to study manganese and other impor-

tant minerals from the political angle. The reports of these committees are of great value, but as yet they are scarcely more than early gropings in a partly unfamiliar field, which have by no means been carried far enough to give a firm basis for sound public opinion.

MANGANESE AND TARIFFS

It might even be suggested that a clear understanding of the world manganese perspective also might afford a more intelligent approach to the tariff problem. This is a complicated and politically dangerous question which I shall touch very lightly at one point. As I see it, no fundamental objection can be raised to import or export tariffs on manganese as revenue producers, or as a means to equalize living conditions in the industry, or to increase the efficiency of transportation by causing home conversion into alloys, or to promote home exploration and development in the hope of finding adequate supplies and to shelter the industry in its earlier stages. When export tariffs are made too high, as happened recently in Brazil, other sources are drawn on. But when, as in this country, the purpose of the tariff is not primarily to produce revenue, and it has been established beyond reasonable doubt that supplies are too limited to play any large part in the manganese industry, or that, such as they are, they should be conserved for times of national emergency, the tariff becomes merely a nuisance to the consuming industry, without corresponding gain to any one else, and hastens the already too early date of exhaustion of domestic supplies.

Parenthetically it may be remarked that there are other mineral tariffs now in force in this country with almost as little justification. Most of them can be ascribed in essence to a failure to understand that nature has put certain minerals in some countries and not in others, and that no amount of legislation can change this situation.

MANGANESE AND WAR

Manganese, as an essential ingredient to steel, constitutes an acute war problem, as I need not remind you. For the North Atlantic countries there seem to be several alternative solutions:

First, development of domestic supplies. For the United States this supply is very insufficient, as already indicated, and for other principal consuming countries it is almost nonexistent. It will be still less in the United States if used too liberally in peace times.

Second, substitution of other substances for manganese, a course which in the present state of science has not been found practicable on any considerable scale.

Third, storage. This is possible, desirable and essential as a means of meeting the first shock. In the year preceding the war Germany

had imported over twice its normal consumption of manganese, but it did not prevent acute shortage. Since the war France has imported several times the amount of manganese needed to make the steel produced. The problem is a live one to the war departments of the world. To sustain a long war on the basis of stored manganese seems to be almost an impossibility for the larger countries of the North Atlantic, simply because of the quantity required. To make the United States independent for a war as long as the last one would require the storage of an immense tonnage, the withdrawal of which from the market would undoubtedly materially affect the normal supply and demand, and consequently prices. There seems to be no way of doing it without its being generally known. Also there is the great practical difficulty that it would require a cold-blooded appropriation of large sums of money by Congress in peace times for possible future war, a problem which of course bristles with political difficulties. There is room to help in the creation of the necessary public opinion. But, at best, for a long-continued war, storage is only a partial and temporary expedient.

(I may say that the report of the manganese committee of the Mining and Metallurgical Society combined with that of the American Institute of Mining and Metallurgical Engineers, devotes a considerable amount of attention to this specific point and makes specific recommendations. It is also perfectly clear that our war and navy departments are alive to that phase of the situation. It is equally clear that the war and navy departments of other countries are alive to the situation. It is not equally clear that certain other departments of our government are giving corresponding emphasis or attention to the problems of manganese which affect peace time.)

Fourth, the only really adequate solution seems to be control of the channels of flow from the sources. For our country this means control of sea routes, which are longer and more exposed than those of any other country. England and Continental Europe can draw their supplies from their own back yards, so to speak—Russia, India, and the Sinai Peninsula. Even on the west coast of Africa it is fairly accessible. The United States would have to cover a long exposed route from Brazil or from the west coast of Africa or across the Pacific from India. This problem has an immense number of contingencies depending on the nations involved in the war, and their alliances. I do not attempt its discussion beyond calling attention to the obvious fact that a combination with the British Empire would more nearly assure the safety of our far-reaching supply routes than any other in sight.

Manganese is only one of several important key mineral commodities for which such a political combination would be mutually advantageous, either for peace or war. Together these countries control commercially over 75 per cent. of the world's production of minerals. Each controls

essential supplies that the other lacks, and together they can meet almost any mineral requirement. In peace times their combined action could insure a more orderly development of the world's minerals by fairer methods than are now in some places in vogue. In war within this group each side would be heavily handicapped. Standing together, there is no other possible combination as strong.

There are of course many other factors, political and commercial, involved in any question of a foreign alliance. I do not make this suggestion, to be entirely frank, with any immediate hope that anything of that sort is going to develop. I merely am pointing out to you the possible bearing of mineral distribution on political relations. I think you will agree with me that it is a potent one.

DISCUSSION

R. C. ALLEN, Cleveland, Ohio.—This meeting¹ was called to consider the very important question of manganese. However, the central thought which is developed in the program runs far beyond the one essential mineral—manganese. We are here considering a group of problems, all of them alike in character but differing somewhat in detail.

Judging from the letters which have been coming through the general committee on program, there seems to have been in the minds of many that this meeting was called for the sole purpose of considering the tariff on manganese and particularly was it called in the interests of the so-called iron and steel group. I would like to take this opportunity to dispel that idea. The American Institute of Mining and Metallurgical Engineers is not a political body and it is not the purpose of this meeting to blow the lid off the tariff.

However, one cannot discuss a subject such as this and at the same time exclude one of its most important angles. There is no reason in the world why the public policy with respect to manganese should not be discussed openly and frankly in a meeting such as this. In so doing, I think I should state on behalf of the Institute, its officers and directors, for fear that somebody else does not, that the Institute takes no position, as an Institute, on matters such as the tariff on manganese and other minerals.

Presumably, the tariff on manganese was placed there for the purpose of developing in this country supplies of manganese ores sufficient to make us independent at least partly, and particularly to make us safe in times of war. There are men here today who were very close to the manganese problem during the war, particularly Dr. Leith, who can tell you of the great difficulties during the great war in getting supplies of manganese to support the iron and steel industries of this country.

The tariff on manganese has stood for about five years. If my memory serves me correctly, the production of domestic manganese, instead of going up, has been going down. There were many of us who believed at that time that there is not in this country a sufficient supply of manganese to support an industry. That was the position taken by those departments of the government whose duty it is to investigate sources of production in this and other countries, and many of us believe they were right.

¹ Meeting held at Cleveland, Ohio, April 19-20, 1927, under auspices of Ohio Section and the Iron and Steel Committee; special topic, Resources and Metallurgy of Manganese Ores.

Whether those deposits exist or do not exist (and we are assured by one of the ablest committees that has ever analyzed any problem) that they do not exist it is evident that five years of high tariff on this essential commodity has not produced much manganese in this country. It is not here.

There is another problem which has been touched upon by Dr. Leith which is to be considered wholly apart from the tariff. That is a general preparedness program in times of peace. There are a number of these essential minerals which do not occur in our country in sufficient quantities and for which we are wholly dependent, or in great part dependent, on other countries. Manganese is only one of them. We have information that other countries, particularly in Europe, are much more forward looking than we are. They have developed and are executing a policy designed to make them safe in times of war with respect to the essential commodities necessary to carry the country on and furnish the necessary munitions.

As far as the general public knows the United States has no such policy and despite specific recommendations which have been made by non-official bodies, such as the joint committee of the American Institute of Mining and Metallurgical Engineers and the Mining and Metallurgical Society, no attention has been paid to the recommendation of this committee and nothing has been done, as far as the general public knows, in the way of making our country secure in times of war, with respect to manganese and a number of other essential minerals needed.

D. F. HEWERT, Washington, D. C.—During the past year, I had the good fortune to spend five months in Europe. The principal purpose of the visit was to attend the meetings of the International Geological Congress, but a secondary purpose provided for a visit to a number of the classic European mining districts. I gathered considerable data, some of which I hope to put into the records of the Institute in the course of time, but at this meeting I shall only mention one phase of my inquiries and the conclusions that come out of them.

It seems to me, as I meet mining engineers and the other men connected with the mining industry in the United States, that they have a very inadequate perspective of the industry. I wonder how many present here realize that if you look backward less than 100 years, to the period from 1820 to 1860, you will find that England was the principal base-metal mining country in the world and, with regard to current production of the major base metals, occupied a position in the world much the same as that of the United States at present. During recent decades, and especially since the war, production of most of the metals is a very small part of the maxima and a negligible part of the world's output. England's position, therefore, has greatly changed, not only relatively, but absolutely, in less than 100 years.

With regard to copper, England produced from 1820 to 1830, 45 per cent. of the world's output. The peak of production came in 1853 but recently current production has been only a few per cent. of that peak. Scarcely 100 years ago, the lead production of England was nearly half that of the world; the peak was passed in 1856 and recently, it has ranged from 3 to 10 per cent. of that peak. Similarly with regard to tin, Cornwall was the world's principal source 100 years ago; the peak of production was passed in 1871 and recently it has ranged from a third to a half of the peak. Early in this 100-year period, England produced about two-thirds of the iron of the world; the peak of iron ore production was reached in 1880 and recently production is scarcely half that peak.

If you will look into British literature concerning mining, the current press, textbooks and summaries of mining of the period 1860 to 1890—I refer you particularly to Sir Robert Hunts' "British Mining," published in 1883—you will find there much the same sort of debate that is taking place in various places in the United States and in this meeting. The conclusion reached by many at that time was what

we are now learning, that mineral deposits are exhaustible, and as human affairs go, rather quickly exhausted. The debate then, as now, was how they might maintain mineral supplies for the critical industries, whether by placing import tariffs on the minerals, thus sustaining the domestic industries by raising the price of the product, or by advances in technique, thereby making lower grades available, or by going out into other parts of the world and acquiring needed supplies of the desired grades.

I am not a close student of British tariff history, but I have received from good authority at Washington, the information that following the Napoleonic Wars, Great Britain entered a period of widespread protection to domestic industries. Later, through what successive steps and for what reasons, I cannot attempt to present here, there arose a decided feeling toward free trade with the result that, in the period between 1850 and 1865, the Kingdom gradually adopted a free trade policy. The only deviation from that policy has taken place during post-war years, in the form of the Safeguarding of Industries Act, and that, I am informed, has not touched the raw materials.

What conclusions may be drawn from British history in these matters? It seems to me that the United States, if I read the trend of production correctly, is going to pass through the peaks of domestic production of the principal metals, at present price standards, during the next 10 or 20 years. The peaks for some metals assuredly have been passed already. How is the country going to maintain the supplies of metals for the existing industries? I see no other way than those considered by Great Britain. That country did not try the method of protection, but faced the situation in its broadest aspects and hunted for materials throughout the rest of the world.

Doubtless, advances in technique will tend to sustain production by discoveries of some new sources and by making lower grades of ore available. Broadly, it seems to me that the United States has no other course open in the near future than to look forward to depending more and more upon foreign sources for many of the principal metals. At least, we are fortunate in having before us the record of the production and length of life of many British metal mines and the effect of governmental policies on them.

J. S. GRASTY, Charlottesville, Va.—It seems to me that while it is disavowed that the question before us is a political one, that unquestionably it is a very large political question because it touches on the matter of tariff and the matter of internationalization of minerals. It seems that it is desired to make a league of nations, so to speak, as applicable to certain minerals.

The very nature of the occurrence of manganese in this country is such that it takes time and money to bring about development, and while time has been had to some extent, owing to the tariff protection, probably not enough money has been had. At any rate, I think we should realize that we are projecting ourselves into a political question without any doubt. That is true no matter how much we may disavow it.

As to the matter of how much tonnage there may be, I think it is not accepted universally that the committee's conclusions are absolutely correct. I, myself, have just finished the work of investigating manganese occurrences in Virginia of a type which has not been described as far as I know anywhere, except briefly, nor has it been investigated by this committee. It is a replacement type of occurrence, the same type as the Oriskany iron ores. It is a very unusual thing for manganese in quantity, and is found along the eastern front of the Alleghenies and northwest of Woodstock as well as elsewhere. It has been developed to the north of Woodstock by the Hygrade Manganese Co., and there is a very large deposit indeed on what is known as Mineral Ridge. The owners of this company have not undertaken so much to sell ore as they have to develop and block it out. I think the whole matter calls for more study and no quick conclusion should be reached.

C. K. LEITH.—How much has been blocked out?

J. S. GRASTY.—They have blocked out as positive ore approximately 300,000 tons. It is estimated, on the basis of the occurrence of the deposit that they have as probable and possible ore a tonnage amounting to about 700,000 additional tons.

C. K. LEITH.—Of what grade?

J. S. GRASTY.—This ore will average about 25 to 30 per cent. metallic manganese and calls for concentration. They are concentrating ore now and are milling it most successfully. They have devoted 8 years to development and plan to block out the ore and develop it on an even larger scale. There are other occurrences in the same alignment, that is, to northeast and the southwest. As to tonnage, it is a very unusual occurrence which any geologist, or engineer, would be very much impressed with.

C. K. LEITH.—I take it for granted that the engineers who constitute various committees would be the first to disclaim any finality for their figures. The best anyone can do is to make estimates and judgments and then hope for light on the situation later.

A. S. DWIGHT, New York, N. Y.—My chief interest in the subject under discussion is to emphasize the great importance of the united effort to protect these supplies of manganese and other strategic metals for the possible use of this country in times of war.

It is very gratifying to me, and I am sure it is to the other members of the Committee on Industrial Preparedness to see the great importance which is now attached to this report of our subcommittee on manganese. The only credit which I can claim is in helping start the organization of the committee during the year that I had the honor to be president of the Institute and to help select the very able men who made up that committee and who were induced to give the time and thought and expression of their own experience along these lines for the benefit of the government.

As far as the Institute is concerned, the movement was initiated by the Assistant Secretary of War who asked for the cooperation of the Institute in helping solve some of these problems which were imposed upon his office under the National Defense Act. The Assistant Secretary is charged with the duty of procurement of all supplies necessary for the maintenance of military operations in time of war.

In pursuance of this request a Committee on Industrial Preparedness was appointed which undertook to engage the interest and services of men in the membership of the Institute who were experts in the various metals concerned. About the same time the Mining and Metallurgical Society of America was starting a series of studies along a nearly parallel line, and, so to avoid duplication of effort, an arrangement was made whereby the subcommittees for the various metals were appointed jointly. These subcommittees prepared reports of great value, covering the following 10 so-called strategic minerals and metals, viz., antimony, chrome, graphite, manganese, mercury, petroleum, platinum metals, tin, tungsten and vanadium. The only variation in the reports of the two Societies was in the case of manganese. The A. I. M. E. report was longer and dealt more specifically with the problem of national defense, urging upon the Government that an emergency stock of 600,000 tons of high-grade manganese ore should be accumulated and held as a safeguard to the iron and steel industry in case of an interruption of foreign imports in time of war. This report has received wide and favorable consideration, and is apparently to be the main basis of the discussion at this meeting.

In closing, I will say that while our general committee on industrial preparedness has for the present completed its task, its existence has been continued by sub-

sequent presidents, as an emergency committee, ready to take up any question in connection with national preparedness which the War Department may wish to present for study.

D. F. HEWETT.—Mr. Grasty said that a type of deposit not considered by the Committee has been developed in Virginia. I want to be sure that I understood him correctly. The report by Stose and Miser,² involving about 8 months of field work along the west side of the Valley of Virginia, and published by the Virginia Geological Survey, describes many deposits in the Oriskany sandstone zone. One type replaces the sandstone and appears to be bedded locally, whereas the other forms large, tabular masses on the surface where the Oriskany zone meets peneplains. Am I to understand that there is something along the Oriskany zone, entirely different from these?

I would like to ask Mr. Grasty, if he cares to state it, at what price per ton he thinks that ore could be produced from these deposits at a profit.

F. L. HESS, Washington, D. C.—We are immensely interested in any new resources and if this ore has to be concentrated, as I understand it does, the question of the amount that can be saved through concentration immediately becomes important. You say that it is 30 per cent. ore and it must be concentrated considerably to make it valuable. There must be a loss in concentration or else it must be a new type of concentration.

You state it is a replacement. I have done much work on them but have never seen a replacement deposit where a man could foretell one foot ahead where it was going. I would like to know on what data you suppose there is 700,000 tons, if it is a replacement deposit.

J. S. GRASTY.—I want to make the answer brief. In the first place, I did not intend to convey the thought (and I am sorry if I did) that the U. S. Geological Survey was not acquainted with the replacement deposits of manganese. I know Mr. Miser and Mr. Stose have been in that territory. I also know that since Mr. Miser was in that territory a vast amount of development has occurred, that where the orebody in the replacement zone had been driven on a few hundred feet it has been developed many times that since then. Therefore, Mr. Miser's conclusions were based upon what he saw. He would revise his opinion if he could see it now.

As to the development, the property has been developed 3000 ft. along the outcrop and by drifts and crosscuts in the ridge so that the underground workings total about 1500 to 2000 ft. and this has been strictly (at least 90 per cent. of it) development work. The men operating this property have put their money back into blocking out the ore. The formation is folded in such a way that there is a large deposit on the side of an anticline with many minor folds on the flank of this anticline.

In other words, it is like the type of other Oriskany occurrences in Virginia where you find large deposits of iron ore. Another thing, usually in these so-called Oriskany occurrences the ore is not all "Oriskany." These gentlemen have done no development into that part of it below the Oriskany. In view of the nature of the folding and fracturing it would pay them to investigate further and probably block out in larger tonnage in older limestone below the Oriskany horizon.

As to the beneficiation and concentration of the ores, I know that it is not claimed they will get a 100 per cent. return on that, but they show that they can get 80 per cent. or more. I hope I have in this brief way been able to answer the questions, but time permitted I would be glad to discuss this occurrence more in detail.

²G. W. Stose and H. D. Miser: *Manganese Deposits of Western Virginia*. Bull. No. 23, Virginia Geol. Survey (1922).

C. K. LEITH.—I think it is apparent from the discussion and from such contact as I have had with questions of this sort through a series of years that our normal interest in minerals, as far as it touches the political aspects at all, runs to questions of tariff. We think of tariff and conservation; we think of the question of preparation for war—all of those questions are of course live ones. They are being taken care of. They are so necessary that we must meet them. The more I see of the situation the more I feel that the one great part of the field not now adequately covered by thought in our own group is that related to foreign exploitation.

I imagine if some members of the State or Commerce Departments were here they could bring up questions illustrative of the sort of thing that is before them day after day relating to the question of mineral exploitation. As I see the question discussed it seems to me there is more lack of information as to what is going on in that field, as important as it is, than in most other aspects of the mineral problem. I come from a section of the country where I venture to say if you asked for a vote of opinion that 99 people out of 100 would deny that we are exploiting any minerals in Mexico, or have any need to, or if we are that perhaps we had better stop. We do not see anywhere in the public press any recognition of the necessity for this exploitation—even when it applies to as clear a case as manganese. The whole question of the open door for mineral development and exploitation needs defining. It can be defined and is now being defined by people outside of our profession. Should we not participate. That is one of the things I hope to bring home to this group.

[Note.—Following the Cleveland Meeting and the discussion there, the War Department requested the Industrial Preparedness Committee of the Institute to reconstitute its special committee on manganese to consider what, if any, revision should now be made in its estimates and recommendations. This is now being done.—SECRETARY.]

Manganese Resources in Relation to Domestic Consumption

JOHN V. W. REYNOLDERS, NEW YORK, N. Y.

(Cleveland Meeting, April, 1927)

OUR entry into the World War suddenly brought home to us in a startling way the vital importance of manganese. Since the war, much has been written and said upon the subject of manganese and a great deal of time and money have been spent in attempts to find an adequate substitute for this metal in the manufacture of steel; none has been found. It is true that ways are known by which it is possible to make small amounts of good steel without manganese; but taking into consideration the diminishing quality of iron ores and fuels as well as the increasing exaction of specifications for finished products, it may safely be assumed that high-grade manganese ore will be more and more a necessity in the production of steel as time goes on. Taking world resources as a whole it may be well to point out that the problem of obtaining needed supplies of manganese from a strictly commercial viewpoint is now less serious than it was before the war. Formerly it was feared that world resources of high-grade manganese ore might be exhausted in a relatively short time, but, largely as a result of the stress laid upon the importance of manganese during the war, the world's available supply of high-grade manganese ore has been greatly augmented by new discoveries and fuller development of previously known deposits.

SOURCES OF PRODUCTION

Russia alone, of countries producing the major part of the world's steel, has within its geographical boundaries sufficient manganese to supply its needs. The domestic production of England is inadequate and English requirements for high-grade manganese are supplied principally from India. France also draws largely upon India, as it depends upon foreign sources of supply except with respect to a small tonnage of silico-manganese and silico-spiegel produced in the Hautes-Pyrénées. Germany and the United States likewise depend upon foreign sources of high-grade ore, though each possesses comparatively large reserves of ferruginous manganese ore and mangiferous iron ore.

At present the world demand for high-grade manganese ore is met with ease by the production of Russia, India, Gold Coast and Brazil.

Russia

It is probable that Russia produced in 1926 approximately 831,000 tons of high-grade manganese ore derived from the Tchiaturi deposit of Georgia and the Nikopol deposits of the Ukraine. In addition to these deposits, Russia has large undeveloped resources which will become available when they are needed.

Georgia.—Of all known manganese deposits in the world the Tchiaturi deposit in Georgia is the largest. Estimates of its reserves vary from 50,000,000 to upwards of 100,000,000 tons, the lower figure being more nearly in accordance with the facts. The orebodies are included in an area of about 22 sq. miles. The bed of ore varies from 5 to 8 ft. in thickness. In quantity, quality and availability of ore, this deposit is so pre-eminent that it may be expected to supply for many years a large portion of the world's demand for manganese, though the losses entailed by mining and beneficiation, as practiced in the past, were so exceedingly wasteful as to hasten the exhaustion of the deposit. Shipments from this deposit in 1926 probably amounted to about 396,000 tons.

The Ukraine.—The reserves of the Nikopol deposits have been variously estimated at from 7,000,000 to 10,000,000 tons, although the known extent of exploration would class much of this as only "possible." Prior to 1920 practically all the Nikopol ores were utilized in the manufacture of steel in Russia. Of late, Germany has been drawing largely upon these deposits for her needs. It is claimed that Nikopol produced 435,000 tons of beneficiated manganese ore in 1926, and Russian publications have stated that 235,000 tons were exported to countries other than Russia and Germany. The high silica content of these ores renders them less desirable than those of Georgia.

India

There are many known deposits in India containing large reserves of commercial ore. The production of India has been relatively constant for many years and exhaustion seems to be far in the future. In 1926 India's estimated production was about 800,000 tons of ore and her exports amounted to about 700,000 tons. That year was marked by a consolidation of the largest producers of Indian ore, the Central Provinces Manganese Ore Co., Ltd., with important English manufacturers of ferromanganese in a company known as the Kingdom Ferro Manganese Co., Ltd. This will assure a market for the largest producer of Indian ore and give to British manufacturers direct control of the source of their raw material.

Gold Coast, Africa

The Gold Coast deposits are known as the Dagwin Mines. They are situated 33 miles from the Port of Sekondi, with which they are connected

by rail. Development work to date has proved reserves of 10,000,000 tons, and future development is expected to more than double this amount. The ease with which the deposit may be mined, and the abundance of cheap labor available, indicate that this deposit can meet the low costs of the Tchiaturi deposit without difficulty. Port facilities having been improved and lighterage done away with, the estimated production for 1926 was 360,000 tons. The output of this deposit is said to be controlled by American interests, though the larger part of it goes to Norway, and a considerable portion to Canada, in which countries it is manufactured into ferromanganese, while the balance is shipped as crude ore to the United States, France and England.

Brazil

The total exports of Brazil up to and including 1925, slightly exceeded 6,000,000 tons. Ninety-five per cent. of this amount came from two districts—the Nazareth district in the State of Bahia, and the Lafayette district in the State of Minas Geraes. It has been estimated that reserves in excess of 15,000,000 tons are contained in Brazilian deposits now accessible to transportation, although the most important, those of Minas Geraes, are more than 300 miles distant from tidewater; but Brazil is known to have, in addition, extremely large resources of manganese in deposits at present inaccessible to transportation. The output of 1926 will be greater than that of 1925—311,882 metric tons. The United States consumes the greater portion of the Brazilian production, the largest developed deposit being owned by the United States Steel Corp. The balance of Brazilian ore is exported chiefly to Belgium, France, Great Britain and Holland.

Other Resources

Other large deposits are known in Australia, Dutch East Indies, Chile and Cuba, which at favorable prices will ship some ore. There are persistent reports that the deposits of high-grade manganese recently discovered in the Postmasburg District in the Bechuanaland Protectorate of South Africa contain reserves of such magnitude as to eclipse all other known deposits; but authoritative confirmation of this is lacking as yet.

RÉSUMÉ OF RESOURCES

The low cost of production from the four great sources, Russia, India, Gold Coast and Brazil, the magnitude of their reserves, their accessibility to water transportation and their ability to expand production readily, are the factors which tend to make the mining of manganese in other sections of the world non-commercial. As an example, Cuban production was relatively constant for some years until 1926, when a

drop in price caused cessation of shipments. Chile likewise is a marginal producer. It would seem probable that deposits of manganese other than those mentioned will be forced to remain dormant until economic exhaustion is more nearly approached by the four great sources. It must be taken into consideration that with the possible exception of the Gold Coast a combination of two of any of the three great sources could with comparative ease supply the world's present demands of relatively 2,400,000 tons.

Table 1¹ presents the statistics of importation in detail:

TABLE 1.—*Manganese Alloys and Ore Produced in the United States and Imported from Principal Foreign Sources, 1917–1925, in Gross Tons*

Year	Steel (Production) ¹	Ferromanganese		Spiegeleisen		Manganese Ore				
		Shipments	Imports	Shipments	Imports	Shipments ²	Total Imports	Imports from		
								Russia	India	Brazil
1917	45,060,607	260,225	41,969	189,241	3,968	129,351	629,972	48,975	512,517
1918	44,462,432	306,076	27,168	263,861	1,969	305,869	491,303	29,275	345,877
1919	34,671,232	198,255	33,022	90,855	27	54,957	333,344	6,916	9,200	246,592
1920	42,132,934	276,594	59,254	103,672	5,234	94,420	599,764	18,762	71,238	421,523
1921	19,783,797	111,374	9,077	69,230	307	13,531	392,606	113,730	262,468
1922	35,602,926	161,612	410,725	70,253	8,151	13,404	425,000	524,250	234,598
								1,642	298	440,595
1923	44,943,696	226,365	4113,833	134,086	4,668	31,500	419,000	711,670	725,568	788,036
1924	37,931,939	232,821	456,588	76,179	7,433	56,515	4505,000	741,097	754,566	754,938
1925	45,393,524	254,005	478,713	95,890	1,090	98,324	4615,000	7114,537	723,504	7103,650

¹ According to American Iron and Steel Institute.

² Exclusive of small quantities of ore shipped for use in fluxing.

³ Exclusive of 7173 tons in 1920 and 8748 tons in 1921 reported by Bureau of Foreign and Domestic Commerce as manganese ore. The value of this "ore" ranged from \$46 to \$292 a ton; consequently the Bureau of Mines believes that the figures represent imports of ferromanganese, spiegeleisen, manganese chemicals, and perhaps some chemical ore.

⁴ Prior to Sept. 22, 1922, the Bureau of Foreign and Domestic Commerce reported the gross weight of manganese ore and ferromanganese imported, but since that date has reported only the manganese content. This figure is a Bureau of Mines estimate of the gross weight of ore or alloy imported.

⁵ Jan. 1 to Sept. 21, 1922. Figures represent gross weight of manganese ore imported.

⁶ Sept. 22 to Dec. 31, 1922. Figures represent manganese content of ore imported.

⁷ Figures represent manganese content of ore imported.

POSITION OF THE UNITED STATES

The position of the United States is clearly that of a consumer, as the known domestic resources of high-grade ore would not supply our present requirements for 2 years, even if all of the ore could be mined in so brief a period. In the event of a national emergency, which cuts off present sources of supply in whole or in part, no solution is possible

¹ Compiled by the Bureau of Foreign and Domestic Commerce, U. S. Department of Commerce.

save through the accumulation in time of peace of a reserve of foreign ores sufficient to tide over the needs of the army and navy covering the period during which the normal avenues of trade may be closed or subjected to interference.

It is unlikely that a greater urgency to discover substitutes for manganese will ever be experienced than was the case during the World War. Germany, at the beginning of hostilities, whether by lucky accident or design, had stored within its borders exceptionally large quantities of manganese ore. Nevertheless, the unlooked for duration of the war forced German steelmakers to great lengths in locating materials to take the place of manganese. That no satisfactory solution was found is evident from the immediate resumption of the use of manganese as soon as the channels of trade had been reopened.

The hope that the beneficiation of the manganiferous iron ores and ferruginous manganese ores, whereby the metallic manganese content would be made available for the manufacture of high-grade steel, is a futile one in that the time element involved in making our known reserves of these ores available would be prohibitive. Limited independence of foreign supplies might be attained provided a solution were found for the metallurgical problems involved in the treatment of the Montana rhodonite and rhodochrosite and the bementite of the Olympic Peninsula of Washington.

Tariff

The production of domestic manganese ore since the enactment of the Tariff Act of 1922 illustrates clearly and beyond dispute the erroneous theory upon which it was based. With the exception of a short interval represented by the years 1885 to 1890, when the Crimora mine was at the height of its production, the manufacturers of steel in the United States have had to depend upon foreign sources for at least 85 per cent. of their needs. The exception to this statement is found in the production from domestic mines during the years 1917 and 1918, when upwards of 32 per cent. of the country's requirements were supplied by domestic producers. The fallacy of accepting this war-time production as indicative of the ability of domestic mines to supply the country's needs in times of peace is apparent when it is realized that, in order to utilize domestic production, requirements as to analysis of ores had to be greatly modified and domestic production was supplemented by the importation of more than twice as much high-grade Brazilian ore.

A study of Fig. 1² indicates that in order to maintain the production attained in the early part of 1918 a material increase in price would have

² Report on Manganese for U. S. War Department. A. I. M. E. pamphlet published in 1924. (Report prepared by Sub-Committee on Manganese, A. I. M. E. Committee on Industrial Preparedness; see page 62, International Control of Minerals.)

been necessary. It seems a fair inference that if the conditions existing in the latter part of 1918 had been prolonged for a period of, say 6 months, in spite of the importation from Brazil, modifications as to grade of steel produced would have been inevitable, and a repetition, in some degree, of Germany's experience of 1917 would have been the outcome.

The result, as you know, of the domestic production during the World War was the passage of the War Mineral Relief Act, under which 461 manganese claims for relief were filed. Ten properties contributed 52 per cent. of the 435,220 tons produced in 1917 and 1918. It is of more than passing interest to note that of these 10, four were financial failures and one was physically exhausted by this production. Also emphasis must be placed upon the fact that 461 claims were filed by the 420 pro-

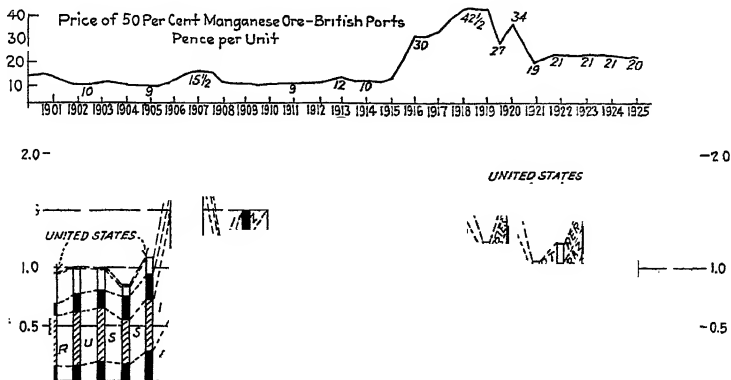


FIG. 1.—PRODUCTION OF MANGANESE ORE—INDIA, RUSSIA, BRAZIL, UNITED STATES AND REST OF WORLD, 1901–1925.

ducing properties, possibly showing that some mines at least were failures under different ownerships.

During the campaign inaugurated by some of the war-time producers of manganese, which led up to the passage of the 1922 Tariff Act, the argument used was, in the main, "national defense," the theory being that in times of duress the industry could be rapidly expanded from a going concern, thus reducing the time that would be necessary to develop our domestic resources. In other words, in the face of the well known limited extent and adaptability of our reserves it was proposed to put a premium on their exhaustion on the plea of establishing an industry that would be readily at hand to function promptly in time of war. The fallacy of this course of reasoning need scarcely be pointed out. Finally Congress was induced to enact a tariff in which the metallic content of manganese ore, in excess of 30 per cent., was made dutiable at 1 c. per lb. and the manganese content in ferromanganese at 1⅞ c. per lb. Broadly speaking, during the years since the passage of the tariff, the world's

price of manganese ore c.i.f. Atlantic seaboard has been 40 c. a unit, or on the basis of a 50 per cent. ore, \$20 a ton, to which must be added, before this material enters into domestic consumption, a duty of 1 c. per lb., or \$11.20 a ton, slightly in excess of 50 per cent. of its value in the world market.

TABLE 2.—*Indicated Total Reasonably Possible Reserves of Manganese Ore of Ferro Grade (35 Per Cent. or More of Manganese), at an Index Price of \$50 a Ton, in Gross Tons, by States*

	Crude		Concentrate		Total		Recoverable Manganese	
	Minimum	Maximum	Minimum	Maximum	Minimum	Maximum	Minimum	Maximum
Alabama.....			5,000	10,000	5,000	10,000	2,250	4,500
Arizona.....	60,000	80,000	15,000	20,000	75,000	100,000	33,800	45,000
Arkansas:								
Batesville.....	160,000	210,000	125,000	165,000	285,000	375,000	132,500	174,300
Western.....	600	1,000			600	1,000	300	500
	160,600	211,000	125,000	165,000	285,600	376,000	132,800	174,800
California.....	100,000	160,000			100,000	160,000	40,000	64,000
Colorado.....	150,000	250,000			150,000	250,000	52,500	87,500
Georgia.....			85,000	200,000	85,000	200,000	35,700	84,000
Montana:								
Butte, carbonate...	135,000	450,000			135,000	450,000	48,600	162,000
Butte, low-grade...			75,000	125,000	75,000	125,000	31,500	52,500
Philipsburg.....	270,000	315,000	70,000	140,000	340,000	455,000	136,000	182,000
Other.....	400	400			400	400	150	150
	405,400	765,400	145,000	265,000	550,400	1,030,400	216,250	396,650
Nevada.....	20,000	30,000			20,000	30,000	8,000	12,000
New Mexico.....	13,500	20,000			13,500	20,000	5,400	8,000
Oregon.....	1,000	2,000	27,000	50,000	28,000	52,000	11,200	20,800
Tennessee.....	3,000	5,000	37,000	55,000	40,000	60,000	15,200	22,800
Utah.....	9,000	12,000			9,000	12,000	4,050	5,400
Virginia.....			300,000	1,000,000	300,000	1,000,000	126,000	420,000
Others*.....	8,000	10,000			8,000	10,000	3,360	4,200
	930,500	1,545,000	739,000	1,765,000	1,669,500	3,310,400	686,510	1,349,650
Chemical ore.....	270,000	315,000			270,000	315,000	108,000	126,000
	660,500	1,230,400	739,000	1,765,000	1,399,500	2,995,400	578,510	1,223,650

* Idaho, Maryland, New Jersey, North Carolina, Oklahoma, Texas, Wyoming.

Possibly the situation can be more tersely expressed by calling attention to the imports of high-grade manganese ore during the past year. In 1926, upwards of 700,000 tons of metallurgical ore were imported. Meanwhile, the domestic production of metallurgical ore was but 22,400 tons, thus showing that only 3 per cent. of the consumption was supplied by domestic mines. Congress in 1922 was led to believe that with the passage of the act referred to, the domestic mines would produce an appreciable amount of our needs, and might render us independent of foreign exports.

In 1926, on the basis of the imports mentioned, nearly \$8,000,000 was paid to the United States customs by the American consumers of manganese in order that the domestic production might be 22,400 tons; thus, in order to make possible the domestic production of 22,400 tons of manganese ore having a sales value of \$31 per ton, duty included, the public at large was taxed to the extent of \$360 per ton, an outlay of \$10 to produce \$1.

Chemical Ores

The reserves of chemical ore attributed to the United States by the report of the American Institute of Mining and Metallurgical Engineers, previously referred to, were given as 300,000 tons. Since this report was made, these reserves have been drawn on to the extent of 97,534 tons. Recent developments in the Trout mine of the Philipsburg district, Montana, have resulted in the discovery of additional reserves and will, no doubt, increase the life of this producing area for a few years. The present per annum consumption of chemical ores may be taken as 50,000 tons, of which some 20,000 tons were of domestic origin in 1926.

Reserves of Domestic Ore

Setting aside all other phases of the question, the seriousness of the situation may be appreciated by comparing the reserves as estimated by the Sub-Committee on Manganese as existent in 1922 (Table 2) with a similar table (Table 3) published in the 1925 Chapter of the Mineral Resources of the United States, pages 178 to 181, inclusive. Since the passage of the tariff act, 146,209 tons of metallurgical ore have been produced, thus reducing the available supply to a new danger point. Table 4 shows the manganese and manganiferous ore (exclusive of fluxing ore) shipped from mines in the United States, 1919 to 1926, inclusive, by states.

Among the many problems that confront the War Department in connection with its requirements of raw materials in time of war, the question of safeguarding our manganese ore supplies is considered to be of the greatest importance. It is my impression that the recommendations suggested in the report of the Sub-Committee on Manganese have been accepted. These recommendations are as follows:

1. The necessary appropriations should be provided for purchase in the open market, in equally yearly quantities over a period of from six to nine years, of a total quantity of not less than 600,000 tons of ferro ores, equal in grade to the accepted foreign standards; to be held in stock at a suitable location for tiding over a war emergency. This quantity is regarded as an irreducible minimum, being based on the assumptions that in the event of such emergency it will be supplemented by forced domestic production, and by the customary stocks privately owned by certain large steel companies.

2. The substitution of spiegel and high-manganese pig iron for ferro, and the investigation of possible substitutes of manganese, should be fostered and encouraged

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State	Total Production through 1925 (Including Fluxing Ore and Manganiferous Zinc Residue)		Indicated Total Reasonably Possible Reserves of Manganese Ore of Ferro Grade (35 Per Cent. or More of Manganese), at an Index Price of \$50 a Ton, by States										
	Ore Containing 35 Per Cent. or More of Manganese	Ore Containing 5 to 35 Per Cent. of Manganese	Total Number of Deposits Examined	Crude		Concentrate		Total		Recoverable Manganese			
				Minimum	Maximum	Minimum	Maximum	Minimum	Maximum	Minimum	Maximum		
Alabama.....	3,615	2,688	12										
Arizona.....	45,385	38,020	118		59,200	79,200	2,800	7,800	2,800	7,800	1,300	3,500	
Arkansas:													
Batesville.....	95,937	92,322	203		160,000	210,000	112,000	152,000	272,000	362,000	126,500	168,300	
Western.....	334		49		800	1,000			600	1,000	300	500	
Total Arkansas.....	95,971	92,322	252		160,600	211,000	112,000	152,000	272,600	363,000	126,800	168,800	
California.....													
Colorado.....	78,654	359	365		97,700	157,700							
Georgia.....	28,799	3,088,073	92		141,600	241,600							
Michigan.....	114,442	137,165	109				80,800	195,800	80,800	195,800	33,900	84,600	
Minnesota.....	137	5,149,800	23										
Montana:													
Butte, carbonate.....	185,369		2		70,500	385,500							
Butte, low-grade.....		22,253	38										
Philipsburg.....	423,000	10,044	23		186,300	231,300							
Other.....	428	1,579	21		400	400							
Total Montana.....	608,797	33,876	84		257,200	617,200							
Nevada.....													
New Jersey.....	27,059	461,595	58		17,800	27,800							
New Mexico.....	235	3,259,212	4										
Total New Mexico.....	11,422	237,283	61		11,000	17,500							

by every educational means at the disposal of the government. The proper governmental department should be instructed to undertake an organized campaign in this direction, the necessary funds being provided; not only for the purpose of investigations and demonstration, but also for gathering and disseminating information. An experienced practical metallurgist should be in charge of this work, who should at all times keep his finger on the pulse of the country's requirements.

3. Government experts, with the necessary funds at their disposal, should be required to maintain a perpetual inventory of all domestic manganese resources of whatever grade, at the same time keeping in active touch with foreign sources and

TABLE 4.—*Manganese and Manganiferous Ore (Exclusive of Fluxing Ore) Shipped from Mines in the United States, 1919–1926, by States*

State	Ore Containing 35 Per Cent. or More of Manganese			Ore Containing 10 to 35 Per Cent. of Manganese			Ore Containing 5 to 10 Per Cent. of Manganese		
	Shippers	Gross Tons	Value	Shippers	Gross Tons	Value	Shippers	Gross Tons	Value
1919									
Metallurgical.....	39	34,882	\$993,955	24	211,632	\$965,680	9	112,303	\$352,570
Chemical.....	3	20,075	797,163						
1920									
Metallurgical.....	54	79,593	1,856,692	31	357,279	1,450,518	6	279,687	644,572
Chemical.....	2	14,827	539,543						
1921									
Metallurgical.....	10	2,300	47,116	6	8,439	42,755	3	62,670	147,576
Chemical.....	4	11,231	447,981						
1922									
Metallurgical.....	15	2,898	50,195	9	344,674	1,071,780	6	251,614	632,039
Chemical.....	3	10,506	404,965						
1923									
Metallurgical.....	32	11,060	188,748	41	319,666	1,158,628	12	1,072,457	3,598,327
Chemical.....	4	20,440	686,225						
1924									
Metallurgical:									
Alabama.....	3	1,242	24,954	9	611	6,722			
Arizona.....	1	42	798						
Arkansas.....	6	3,400	60,350	6	7,991	37,170			
California.....	2	850							
Colorado.....	2	5,338	48,042	7	27,058	109,289	1	3,891	
Georgia.....	8	1,093	18,304	19	14,182	73,684	6	6,353	16,558
Michigan.....				1	50,556	157,184	1	39,939	
Minnesota.....				4	159,915	439,132	3	361,527	970,351
Montana.....	3	11,933	123,456						
Nevada.....	3	1,310	13,579	3	2,591	15,997			
New Mexico.....	2	775		2	23,246				
Tennessee.....	2	455	11,611						
Utah.....				2	116	1,010			
Virginia.....	2	440		2	204				
Washington.....	1	5,000	100,000						
Wisconsin.....							1	175,316	606,593
Undistributed....			39,920			89,202			120,441
Total metallurgical.....	35	31,878	441,014	55	286,470	929,390	12	587,026	1,713,943
Chemical:									
Montana.....	3	23,512	816,463						
Virginia.....	1	1,125	50,000						
Total chemical.....	4	24,637	866,463						

developments. This work must go further than a simple estimate of reserves in the ground; full information being constantly sought bearing on stocks in hand, the availability of reserves, the probable acceleration of domestic output in an emergency, the price that would be needed to bring it out quickly, the amounts that could be looked for, and so on; to the end that the domestic situation may at any time be appraised quickly and with reasonable accuracy, not only as to requirements, but also as to resources.

4. Artificial stimuli in times of peace, which, if effective, will simply tend to deplete an already extremely limited reserve of ferro-grade and chemical ore, should be strongly discouraged.

While we regard all the above recommendations as of the greatest importance, considering that no one of them should be overlooked, we desire to call your particular attention to the first.

It is obvious that the existing tariff on manganese should be repealed. It fosters no industry and depletes our too scanty reserves of this essential raw material. A burden of \$8,000,000 placed upon the nation's industry

TABLE 4.—(Continued)

State	Ore Containing 35 Per Cent. or More of Manganese			Ore Containing 10 to 35 Per Cent. of Manganese			Ore Containing 5 to 10 Per Cent. of Manganese		
	Shippers	Gross Tons	Value	Shippers	Gross Tons	Value	Shippers	Gross Tons	Value
1925									
Metallurgical:									
Alabama.....	2	938	12,544	5	298	3,087			
Arizona.....	3	294	5,102						
Arkansas.....	4	3,517	79,335	9	5,076	23,140			
California.....	3	869	17,525						
Colorado.....	1	743	5,127	5	7,352	26,565			
Georgia.....	8	1,649	20,281	20	28,038	133,215	5	7,349	23,397
Massachusetts...	1			1					
Michigan.....				2	98,753				
Minnesota.....				3	77,665	184,121	4	741,409	1,577,301
Montana.....	4	55,659	585,031						
Nevada.....	2	850		1	6,659				
New Mexico.....	2	1,588		2	40,848	153,251			
North Carolina..	2								
Tennessee.....	1	352	8,800	1					
Utah.....	1	50		3	666	5,754	1	496	
Virginia.....	3	1,499	22,981	1	1,800				
Washington.....	1	8,113							
Wisconsin.....							1	404,014	
Undistributed....		52	198,073		97	386,183			1,198,705
Total metallurgical.....	38	76,173	954,799	53	267,252	915,316	11	1,153,268	2,799,403
Chemical:									
Montana.....	3	20,529	825,970						
Virginia.....	1	1,622	77,000						
Total chemical	4	22,151	902,970						
1926 (estimated)									
Metallurgical.....		24,200	332,000		366,500	1,119,000		850,000	1,898,500
Chemical.....		19,800	853,600						

in order to force the production of manganese ore to the extent of only \$600,000 requires no argument to demonstrate its absurdity.

On the contrary the United States Government, in line with the recommendations of the "Committee on Industrial Preparedness" should accumulate over a period of years such a stock of manganese ore, of standard grade and analysis, as will meet the needs of the War and Navy departments in the event that the national security at any time should be placed in jeopardy.

Such a reserve of ore acquired gradually would not in any way affect the world markets and its value in the course of time will increase rather than diminish, in which respect it differs from investments in battle-ships which become obsolete in less than a decade. The market value of the entire manganese ore supply which the Committee recommended be acquired in from 6 to 9 years is \$12,000,000, or but a fraction of the cost of a single capital ship. Its effect would be to safeguard the quality of all steel that might be needed in a period of emergency lasting 1 or 2 years.

An ingenious plan has been suggested for acquiring a government supply of manganese ore by requiring that the import duty be partly paid "in kind," in other words, that for every ton of ore brought in a certain percentage be turned over to the government and stocked on government reservations or held in government warehouses in convenient relation to points of ultimate consumption, the necessary equipment and personnel being furnished by the Army and Navy. The present rate of import duty, being approximately 50 per cent., would result in too sudden an acquisition of the needed reserves, but a moderate rate of 10 per cent of the imported tonnage would not disturb the market and would result in the accumulation of a safe ore supply in the course of 10 years.

This would constitute a "painless" way of meeting the situation as far as the War and Navy departments are concerned. It would not cure the evil inherent in an import duty so indefensible and opposed to the public interest as that now in effect on manganese ore.

The executive departments of our government are debarred from instigating or attempting to influence legislation. The military branches may submit recommendations but are not permitted to urge the necessary legislation to carry out such recommendations except through very narrow and well defined channels.

It would, therefore, seem to devolve upon the public, of which the American Institute of Mining and Metallurgical Engineers forms a modest part, to give consideration to the able and thorough report of the Sub-Committee on Manganese and to assist as far as possible in carrying into execution its recommendations, which were the result of so much investigation and research carried on in a spirit of disinterested patriotism.

DISCUSSION

A. G. BETTS, Kinderhook, N. Y. (written discussion).—When a business concern hires an office boy, it takes on the first bright-appearing young chap to apply and pays him the prevailing rate of wages. One of the chief tasks the office boy is supposed to be competent to perform is to clean out the office. The manager of the business is highly competent in many lines, but for some reason lacks the ability to keep the office clean. To the steel man, it would seem that manganese is the office boy, and it is purchased just as found and offered to the purchasing agent. The steel mill is competent to perform many functions on a grand scale and with wonderful efficiency, but it is unable to make real steel. It is the function of the office boy, manganese, to do this.

It is open to serious doubt if steel mill operators have ever taken seriously the metallurgy of the preparation of crude manganese. It has merely been found that some usable crude manganese may be obtained by running certain manganese ores through an iron furnace when not busy on anything else, or remote from iron ore, or with a lining that is about gone. It is, however, an entirely unwarranted assumption that the metallurgy of manganese has been worked out, or that the supplies of usable raw ore must be limited to materials of some specified analysis. There was a time when ores of copper, lead and zinc, for example, had to be rich and of specified analysis, to be usable, but the commercial raw materials of today bear little resemblance to such ores of the past.

The discussions of manganese that I have seen are based on the assumption that the metallurgy of manganese is fully established and that the only usable manganese raw materials are of the kind used today. These discussions contain another error, not so much of statement, but of implication—that the tonnages of manganese materials in the United States are so much in this state and so much in that. When it requires a period of years and the expenditure of millions of dollars for a competent mining company to explore thoroughly a few acres of mining ground, is it not absurd to say that there are so many hundred tons of manganese ore in a certain state? This fact is not apt to be known fully in 10,000 years. I do not charge the mining engineers who are on record, of making misstatements, because they merely say that so much has been found and is blocked out, but do charge that any assumption that such figures mean anything at all regarding the amount of manganese in the ground is unwarranted.

Steel manufacturers in the country undoubtedly wish to make steel as cheaply as possible and are searching for means to reduce costs, as lower costs mean larger markets. But that the public is similarly interested or that the Government as a measure of national safety is interested in what might be saved the ultimate consumer by reducing the cost of foreign manganese is at least open to serious doubt. The duty on manganese ore increases the cost of an automobile by approximately 20 c.; the cost of a steel railway coach, \$7 or \$8.

It can be questioned whether it is the best national policy to reduce the cost of an automobile, and at the same time give up a source of revenue to the Government, or to foster not only the development of American sources of manganese, but also the development of an improved metallurgy of manganese suited to American conditions.

Manganese might be characterized as a "good thing" from the government standpoint in that it will stand a tax. Should the American Government reduce the import tax on manganese ore, and make it possible for foreign governments to add an equal tax on exports: in other words, transfer this source of revenue to foreign governments? There is no way to induce the Brazilian Government to remove export taxes and cheapen American steel costs. The Brazilian people do not wish to give up this source of revenue, therefore it is proposed that the American people give up the income received from import duties.

As a matter of business the question of import tariff is open to endless argument, but not if it is established that one course promotes national safety and the other detracts therefrom. Our Government at this time is committed to the principle of import taxes to promote American industry, and the duty of manganese is in line with this policy, but the promotion of national safety is and will continue to be the end and aim of all parties worthy of the name, in so far as it is reasonably clear where the path lies.

It is idle to believe that domestic steel manufacturers will do anything particularly to promote American production of manganese. Their business is ruled by competition and it is no more the distinct duty of these manufacturers to spend their money in that way, than for any other class of citizens to do so. It is a proper function of the Government, and a simple and direct means is the imposition of import taxes. Thus, the question resolves itself into one of national policy of preparedness.

I have seen an argument that the present import duty has just the opposite effect: that it causes the rapid exhaustion of our supplies of manganese. I am satisfied that this argument will not appeal to the average mining engineer, whose experience, with mineral development in almost any line, is more likely to be that the more ore is mined, the greater are the commercial reserves in sight. I believe the average mining engineer would rather believe that the more manganese is mined in the United States, the greater will be the usable material in sight and ready for extraction.

I also believe it can be demonstrated that with the duty now in effect, there is now more usable manganese in sight and available within a reasonable time, than there was at the time the duty was imposed, and that if the duty is maintained, it will finally result in this country becoming independent in respect to manganese.

L. B. MILLER, Cleveland, Ohio (written discussion).—The figures given by Mr. Reynders regarding tonnage of high-grade manganese ore occurring naturally in the United States show conclusively why the States do not and cannot respond to substantial production of this type of ore even with the encouragement of a protective tariff; the ore of that type is simply not here.

In regard to his remarks, however, to the effect that the duty on manganese ore is unwarranted and should be repealed, there is a viewpoint that has a bearing on the subject upon which Mr. Reynders has not touched. While the present tariff has had a part in conclusively proving that high-grade ores exist only in minimum quantities, it has nevertheless been indirectly responsible for a development of profound importance. The figures obtainable for good grades of manganese ore for the last five years due to the tariff have stimulated thought and experiment in regard to discovery and beneficiation of ferruginous manganese ores. One outstanding process that has been fanned into life by this five-year period of protective tariff is the Bradley process of leaching. I have commented on this process in discussion of the paper by Joseph, Barrett and Wood.* By this process the highest grades of manganese ore can be produced—much more valuable grades than occur in nature.

This process also proves the value of ferruginous manganese deposits; the next step of investigation on the part of producers of manganese materials would appear to be the development of deposits of ore that will run 15 per cent. or better in manganese, in the shape of oxides or carbonates that are available for improvement by the Bradley process. In searching for high-grade ore, a record of deposits and tonnage of ferruginous manganese ore of 15 per cent. and over in manganese has been relegated to the background.

Carl Zapffe calls attention to deposits of unknown tonnage in Crow Wing County, Minnesota, of the black ore variety, carrying 15 per cent. or better manganese but

*T. L. Joseph, E. P. Barrett and C. E. Wood: Minnesota Manganiferous Iron Ores in Relation to the Iron and Steel Industry. See p. 338.

running from 25 to 35 per cent. in silica. At the present time these ores are of practically no value. Mr. Zapffe does not state what quantity of ore is available in these possible reserves. The intimation is that many millions of tons of ore exist in these deposits.

In Arkansas, work done during the past two years has developed the fact that enormous quantities of ferruginous manganese-bearing residual clay can be and are being brought into commercial use. The total tonnage of these Arkansas reserves is unknown, but it doubtless runs into many millions of tons. This has been touched upon also in my discussion of the paper of Joseph, Barrett and Wood.⁴

There seems little doubt that if search were diligently made for deposits of ferruginous manganese ores, enormous tonnages of this grade of material would be uncovered, thereby making available indirectly through the Bradley leaching process interesting tonnages of the highest possible grade of manganese ore suitable for making ferromanganese, with a probable complete upset of the theory that this country can be supplied with manganese ores only from foreign sources. I do not accept the theory that we cannot get manganese in this country.

The fact that the Bradley process has not yet been put into commercial operation need not detract from the value of the process itself, nor from the results that will hereafter be obtained therefrom. Transformation of ferruginous manganese ores into high-grade manganese ores can be credited unreservedly to the present tariff on manganese ores, as in its absence investigation into the changing of low-grade ore into high-grade ore would undoubtedly have been ignored.

The process of leaching and of beneficiating either the low-grade ores from Minnesota or ore from the residual clays of Arkansas cannot be done at figures that would be generally considered low. The expense of producing these ores and then beneficiating them into high-grade ores is such that some recognition should be granted to the ores and the processes so as to warrant their development and production. Such plans for encouraging the development of ferruginous manganese ore and of the leaching process would appear well warranted. This much is certain, that if the United States is to have a supply of high-grade manganese ore of its own, emanating from within its own borders, such supply must result from some method of improving the grade of ferruginous manganese ore. Ore of this type is relatively plentiful; at least there is no dearth of it, but to this grade of ore comparatively little attention has been paid heretofore.

If it be true that a considerable tonnage of home-grown, high-grade manganese ore, within the borders of the United States, would be considered a welcome national asset, it looks as though a thorough investigation of the Bradley leaching process and the available tonnage of ferruginous manganese ores would be the next logical step rather than to kill off such possible supply by premature recommendation on changes in the rate of tariff.

J. C. ADKERSON, Woodstock, Va. (written discussion).—American manganese mining is in its infancy but is becoming a vital factor in supplying domestic needs, and combined pressure of foreign interests cannot stop the steadily increasing production from American fields. Mr. Reynders gives 1,493,200 tons as the total minimum possible reserve of 35 per cent. plus metallic manganese ore in the United States which can be produced at \$50 per ton index price, but he fails to give weight to the fact that considerably more than 1,493,200 tons of ore of this grade have already been shipped. from American mines at an average price of about \$33 per ton. He says that shipments deplete reserves of the figure named but he fails to show that, while shipment continue, the admitted estimated reserves are increasing year by year and that recent

⁴ T. L. Joseph, E. P. Barrett and C. E. *Op. cit.*, 340.

developments in Virginia and Montana alone have increased the estimated reserves by several hundred thousand tons. The deposits in America have hardly been scratched but they have shipped more than 1,493,200 tons. Any estimate of reserve, until the ore is tested or blocked out, is a guess, but 15 years of engineering research, development and operation in manganese in the United States indicates that it is sensible to estimate that not more than one-tenth of the reserve of the United States has been shipped. This means that the total minimum possible reserve of the United States could better be estimated at 14,932,000 tons.

Mr. Reynders calls attention to the fact that \$31.20 per ton, including duty, is being paid for foreign ore at the ports, but he fails to explain that most American producers are being offered and paid \$5 to \$10 per ton less for the same grade of ore and under the same schedule of analysis. Mr. Reynders gives prominence to the statement that the annual production since the tariff is only 22,400 tons but he fails to account for the record of 1925, which alone shows 98,324 tons shipped from domestic properties during the year.

The tariff follows a firm principle and is justified. It brings to the U. S. Government an income of \$8,000,000 a year and promotes the development of an essential industry which has grown from almost nothing in 1922 to the rate of more than \$3,000,000 per year in 1927. Instead of developing our own reserves, it has been suggested that the government go into foreign fields and buy \$12,000,000 worth of ore and store it away. This carries with it an annual cost of around \$700,000 per year. If we must store up ore for a wartime emergency, let us get it through the development of our own domestic resources and let a portion of the tariff on foreign ores be applied to its purchase.

Furthermore, bear in mind that there have been shipped from mines in the United States 13,741,959 tons of ore containing from 5 to 35 per cent. metallic manganese and that this tonnage is constantly increasing. The total domestic tonnage of this grade of ore has been estimated as 35,000,000 to 48,000,000 tons. There has been over \$200,000 spent in America during the year 1926 alone in developing the beneficiation of these lower grade ores, and large commercial production from this source is now actually under way.

The full and complete records of the shipments and developments of American manganese for the years 1926-28 will help clear the glasses of impartial observers and will serve as convincing proof that America has deposits which will serve a good part in filling the industrial needs of our country.

J. W. FURNESS, Washington, D. C.—In 1923, the revenue received from manganese ore was \$1,659,000; on ferromanganese it was \$3,820,000. In 1924, it was \$4,780,000 on manganese ore, and on ferromanganese it was \$1,800,000. In the last year, 1926, the imported crude ore gave a revenue of \$7,935,000 and the ferromanganese gave \$1,822,000, showing that with the tariff there has been a very decided increase in domestic production and a decrease of foreign production.

A. G. BETTS, Kinderhook, N. Y. (written discussion).—The Institute has a large and able body of members, who are in large degree engaged in the production side of ores and metal products, mostly in this country. The officialdom of the Institute does not, in my opinion, fairly represent the sentiment of the membership, and the great attraction of the Institute is, of course, the ability, character and public spirit of the membership. It seems obvious that the members of the Institute are desirous and willing that their services and abilities be utilized to the fullest extent possible in the growth of exploration, development, mining and metallurgy in this country in order to supply America not only with manganese but also with chrome, nickel, tungsten, clays, bauxites, refractory bases, and many other important products of mines now largely supplied from foreign sources. This rather proves that the

sentiment as a whole would be out of sympathy with the position taken by Mr. Reynders, and it is neither justifiable nor fair to let the impression go forth at the Institute's expense that the Institute is specifically opposed to a tariff on manganese.

What are the obvious business facts today regarding manganese? There are several large foreign producers who are dividing the world markets, and we are told that this country consumes one-third of the manganese. Can it be supposed that these factors are indifferent to anything affecting the future of their market? The matter of this tariff has been rather quiescent for some time. Nobody has been hurt by it, so far. The steel mills have needed manganese, and the tariff has raised the cost of steel to the consumer by possibly as much as 20 c. per ton of steel. The foreign manganese factors have had to pay an equal duty, there was no competing substitute, and a reduction of the tariff would have given no competitive advantage to any of them, or increased their market to any important extent. But if this tariff should result in the development of an American manganese industry, or such a growth could be foreseen as a result of the tariff, would it not be natural for these foreign producers to become more active and demand loudly that the tariff be repealed, before it was too late to destroy an American industry? Therefore, one may ask, is this agitation a tribute to the intrinsic American manganese situation, including our natural resources and the ability and energy of our mining men and metallurgists, and the progress being made by the Bureau of Mines?

Mr. Reynders' demand for the cancellation of this tariff, in such a situation, would be pleasing to foreign manganese factors and to Americans interested in foreign properties. Likewise his recommendation that the Government buy an immense tonnage of foreign ore, and stock it, would also please manganese-ore producers. Even should the Government buy and stock domestic ores at the point of production, it would mean an enlarged market for these factors. However, Mr. Reynders has not so far disclosed any plan for the Government to lay in a big stock of this ore for nothing. Juggling the accounts as suggested will not do. Such an operation, instead of being "painless," would be more likely to give an intelligent citizen a pain.

Mr. Reynders leaves his readers in some doubt as to whether the manganese tariff is a bad thing because it does, or because it does not. He says it is a bad thing because it makes us mine our manganese so fast, and a bad thing because it does not make us mine any to speak of. His argument about the terrible damage done by this tariff "to industry" leaves me in doubt as to where this tariff will be consumed. I think our schoolboys are informed that if the Government gives up a tax in one place, it taxes something else.

F. SAMUEL, Philadelphia, Pa. (written discussion).—I have read with great interest Mr. Reynders' exceptionally able and conclusive paper as to the world's production of manganese ore, which practically covers all that can be said on this. I do feel, however, that something more can be said on the commercial end of the manganese ore business and the placing of duty on manganese ores and the compensating duty on ferromanganese.

✶ This duty was practically insisted on by the member of the Ways and Means Committee from Colorado, who secured a duty on manganese ore that had never existed before and a corresponding duty on ferromanganese on the grounds that the miners of Colorado and Montana should be properly compensated so that the owners of the mines there and other places could compete with the foreign countries far away, mining and shipping manganese ore to this country.

Of course the very effect of this duty, while giving this country a small revenue, is as stated by Mr. Reynders, one of the first causes of depleting the manganese deposits in this country, and undoubtedly there never should have been a duty placed on manganese ore. While depleting the reserves in this country there was placed an

unnecessary burden on every manufacturer of steel or user of manganese ore for chemical purposes. Mr. Reynders was modest in stating that the burden was only \$8,000,000 placed on the steel industry.

The consumption of ferromanganese is in the neighborhood of 350,000 tons a year and the writer feels that the proper way of figuring the cost to the American public is to take the duty on ferro, which, roughly speaking, on the grade of ferro imported to this country, would be \$32 a ton, or in other words approximately \$12,000,000. While of course the large makers of steel produce their own ferro, the price of ferro has increased practically the entire amount of the duty upon its imposition.

Why would it not be possible, instead of purchasing the manganese ore, to purchase the ferromanganese itself to be shipped over a period of years, as not only the handling but the cost of storage and various other matters connected with this matter would be much less than the actual purchase of the ore?

J. V. W. REYNDERS (reply to discussion).—Reviewing the discussion of my paper, I desire first of all to emphasize that I am heartily committed to tariff rates so long as they foster American industry in a reasonable manner. The references made in my paper to the proposed storage of manganese ore as a protective war measure did not originate with me personally, as is assumed by some of those who discussed the paper, but with a joint committee of the American Institute of Mining and Metallurgical Engineers and the Mining and Metallurgical Society of America composed of engineers of wide experience and national reputation, whose recommendations were based upon practical knowledge gained under the stress of war conditions.

The tariff on manganese, the criticism of which seems to have aroused an undue amount of apprehension, became effective in 1923 since which time (excluding the current year) the duties have amounted to about \$30,000,000. In the year 1926 alone the imports on ore yielded \$7,935,000 and on ferromanganese \$1,822,000, a total tax of \$9,757,000. It is reasonable and pertinent to inquire what these vast sums have accomplished in the way of concrete results. My paper is intended to answer these questions in definite terms based on government statistics.

It hardly seems adequate to urge as a defense for the continuance of the present excessive duties that a new process of beneficiation of low-grade ores is in process of development and "intimating" that ores suitable for such process run into "many millions of tons," or that if diligent search were made, enormous tonnages would be uncovered.

The aggregate American production to date is given by Mr. Adkerson as in excess of 1,493,200 tons: I cannot find the supporting statistics for this figure, but find that the production for the 15-year period from 1910 to 1924, excluding the two war years 1917 and 1918, was 323,593 tons and including the war years 765,831 tons. During the same period the rest of the world production was 24,613,585 tons, or 97 per cent. of the total production.⁵ He states that recent developments in Virginia and Montana have increased the estimated reserves by several hundred thousand tons and "that it is sensible to estimate that not more than one-tenth of the reserves of the United States has been shipped." How this particular percentage is arrived at is not stated and such indefinite premises hardly seem an adequate background for a tariff which taxes the consumers of steel to the extent of \$10,000,000 annually, nor does such a tariff seem to bear a reasonable relation to the value of the domestic industry as at present developed which Mr. Adkerson gives as \$3,000,000, although this amount must be manifestly an estimate on his part. The value of the importations in 1926 including duties, as a matter of comparison, was ten times the claimed estimate for 1927 of domestic production.

⁵ C. H. Behre, Jr.: *Manganese*. The Universal Industry (1925) 475. McGraw-Hill Book Co., Inc

Mr. Betts advances the futile suggestion that mining engineers, without regard to the public interests involved, should support a tariff merely because it gives an opportunity to the profession to explore and locate mineral deposits.

In conclusion, I may repeat that I am not opposed to a duty on manganese ore provided that it bears some reasonable relation to the values involved. It is not at all beyond the range of possibilities that a substantial production of manganese concentrates may be brought about in the United States, as suggested in my paper when I referred to the rodonite and rhodochrosite deposits of Montana. An accurate analysis of the production and concentrating costs of these, or any other ores, will readily indicate the specific tariff rates that will enable the enterprise under review to function successfully, with a minimum tax burden upon the consumer. An annual tax of \$10,000,000 imposed in order to support an assumed \$3,000,000 domestic manganese industry seems, to say the least, out of scale.

Minnesota Manganiferous Iron Ores in Relation to the Iron and Steel Industry*

T. L. JOSEPH,† E. P. BARRETT,‡ AND C. E. WOOD,§ MINNEAPOLIS, MINN.

(Cleveland Meeting, April, 1927)

HISTORY OF MANGANESE AS A DEOXIDIZER

THE invention of the Bessemer converter process in 1856 added great impetus to the manufacture of steel and is one of the outstanding contributions to process metallurgy. Although the process of refining pig iron by passing air through the molten metal bears the name of Henry Bessemer, it should be mentioned in passing that William Kelley of Eddysville, Ky., worked on the process as early as 1847. Kelley was granted a patent on the process in 1857, inasmuch as he was able to prove priority in discovery.¹ After considerable litigation in 1865 the respective interests of Kelley and Bessemer were combined.

On Sept. 10, 1856, Robert Mushet of England obtained his first patent on the application of manganese to cast steel. Mushet anticipated trouble in purifying iron by passing currents of air through it. His patent specified that steel made in this way often contained flaws and was red short or cold short. Mushet understood the refining of pig iron into steel well enough to realize the difficulties likely to be encountered from oxidation of the iron.

Mushet was not only an inventor; he was also an investigator. He understood the principle of selective oxidation which takes place in the converter process, and he realized therefore that the remedy for over-oxidation was attainable if a suitable metal could be found at a reasonable cost and in sufficient quantities. His first experiments were made in small crucibles holding a few ounces of metal. Later his work was conducted on a larger scale and with complete success. Manganese had been used in making steel a quarter of a century before the converter

* Published by permission of the Director, U. S. Bureau of Mines.

† Superintendent and metallurgist, North Central Experiment Station, U. S. Bureau of Mines.

‡ Metallurgist, U. S. Bureau of Mines.

§ Assistant chemist, U. S. Bureau of Mines.

¹ R. W. Hunt: *A History of the Bessemer Manufacture in America: Trans.* (1876-77) 5, 201-16.

process was discovered, but Mushet was the first to point out its application to the new method of steel-making.

A great deal of the metal made in the first trials with the Bessemer converter was cold short and red short as Mushet had predicted. This condition was attributed for a time to phosphorus and sulfur. However, when the best Swedish pig iron was used in the Sheffield Works, low-carbon steels were often red short.

Mushet's discovery is one of the most important events in history and yet it is seldom mentioned. It is doubtful whether the steel processes would have developed as they have had Mushet failed to point out early the use of manganese as a deoxidizer. During the 70 years that have elapsed since the invention of the Bessemer process, no metal has been used so extensively as a deoxidizer as has manganese. It is indispensable in the present art of steel-making. High prices paid for manganese alloys during the World War support this contention.

Although the deoxidizing action of ferromanganese is not as strong as some other deoxidizers, it is the most prominent and widely used. Silicon and aluminum, if used excessively, produce non-metallic inclusions and do not eliminate sulfur or render it in less objectionable form as does manganese. Manganese has a marked capacity as a deoxidizer and because of other desirable properties it stands alone and there is at present no satisfactory known substitute for it. The relative merits of deoxidizers depend somewhat upon what is sought in the steel, so that each manufacturer must decide for himself which one will best suit his needs. If deoxidation is to be complete, aluminum, silicon, or titanium are usually used in addition to ferromanganese.

METALLURGICAL USES OF MANGANESE

Approximately 95 per cent. of the total manganese consumed is used in connection with metallurgical operations. Manganese is not only superior as a deoxidizer but adds desirable physical properties to the finished steel. Three grades of manganese alloys are used in the manufacture of steel. The most important of these is ferromanganese (later referred to as ferro), which contains about 80 per cent. manganese, 12 per cent. iron, 6.5 per cent. carbon, 1 per cent. silicon, and from 0.16 to 0.25 per cent. phosphorus. Next in importance is spiegeleisen of approximately the following composition: manganese 20 per cent., iron 74 per cent., carbon 5 per cent., silicon 1.1 per cent., phosphorus 0.10 per cent., manganiferous pig iron, which contains from 4 to 10 per cent. manganese along with the other elements usually found in pig iron. Phosphorus, however, must be low to prevent contamination when this material is added to the finished steel. Two other forms are used to a less extent. These are silicomanganese and silicospiegel. Manganese is alloyed in these cases respectively as follows: silicomanganese, man-

ganese 55 per cent., iron 19 per cent., silicon 25 per cent.; silicospiegel, manganese 22 per cent., iron 65 per cent., silicon 11 per cent.

MANGANESE REQUIRED IN THE STEEL INDUSTRY

It has been pointed out that manganese plays a very important role in the manufacture of steel and that the major part of it (95 per cent.) is used in this way. The function of manganese alloys in making steel is threefold: (1) deoxidation of the steel; (2) control of carbon content commonly known as recarburization; and (3) introduction of suitable amounts of manganese into the steel to improve its quality. In making low-carbon steels it is necessary to use ferromanganese (80 per cent.) in order to keep the carbon within proper limits. Spiegeleisen can be used in the production of rail steel and other material high in carbon. The reason for this is obviously due to the ratio of manganese to carbon in the two alloys. Low-carbon steels will contain more FeO than high-carbon steels and for the same manganese specification in the finished steel will, in general, require more manganese. The fact that more manganese is required in low-carbon steels militates against the use of spiegel in making such steel. In addition to having a desirable ratio of manganese to carbon, ferro has the additional advantage that it does not require melting before use. This melting involves a loss of manganese.

Table 1 shows the relation between steel production, the consumption of ferro and spiegel, and the pounds of manganese required to make a ton of steel during the period 1911 to 1926. The pounds of manganese per ton of steel shown for the various years do not represent the actual quantity used inasmuch as stocks of alloys were carried over from one year to another. However, the average by 3-yr. periods smooths out the larger irregularities and gives the trend of practice regarding the use of manganese. During the 16-yr. period an average of 13.9 lb. of manganese was used in making a ton of steel. If this period may be taken as a criterion of future manganese requirements, it appears that 14 lb. is a safe working figure.

In order to observe the effect which the increased use of manganiferous iron ores has had upon the quantity of manganese required to make a ton of steel, Table 2 and Fig. 1 have been prepared. It will be seen from the curve showing the pounds of manganese per ton of steel by 3-yr. periods that the trend from 1911 to 1922 was to use less manganese per ton of steel. This trend was probably due to the use of basic pig iron containing more manganese and to the high price of ferro alloys which prevailed. From 1922 until the present time the tendency has been to use more manganese per ton of steel. In view of the increasing use of manganiferous iron ores in the production of basic pig iron, such an

TABLE 1.—*Consumption of Manganese¹*

Year	Steel, Tons ²	Ferro				Spiegel				Total Manga- nese Ferro + Spiegel, Tons	Spiegel, Per Cent.		Manganese Per Ton of Steel	
		Do- mestic, Tons	Im- ported, Tons	Total, Tons	Manga- nese, Tons	Do- mestic, Tons	Im- ported, Tons	Total, Tons	Total Manga- nese, Tons		Yearly	By 3 Yr.	Per Cent.	Lb.
1911	23,876,106	74,602	80,263	154,865	123,832	104,013	20,970	124,983	24,997	148,889	16.8	0.628	14.1	14.6
1912	31,251,303	125,378	99,137	224,515	177,740	102,661	1,015	103,676	19,801	197,541	10.0	0.632	14.2	14.6
1913	31,300,874	119,495	128,070	247,565	196,798	106,980	77	107,057	20,860	217,658	9.6	0.694	15.5	15.0
1914	23,513,030	100,731	82,997	183,728	145,806	76,025	2,870	79,496	10,306	162,112	10.0	0.690	15.4	14.4
1915	32,151,036	144,200	55,263	199,523	163,098	114,556	200	114,766	21,149	174,247	12.2	0.542	12.2	14.1
1916	42,773,680	224,103	90,928	315,031	241,370	182,837	3,968	183,200	40,808	278,271	14.6	0.619	13.9	14.4
1917	45,060,607	260,225	41,969	302,194	237,463	189,241	1,969	265,830	51,900	286,453	18.1	0.645	14.5	13.3
1918	44,462,432	306,076	27,168	333,244	234,553	263,861	1,969	265,830	51,900	286,453	18.1	0.645	14.5	13.3
1919	34,671,232	198,255	33,022	231,277	162,251	90,855	27	90,882	17,691	179,042	9.8	0.520	11.6	13.4
1920	42,132,934	276,594	59,264	335,848	241,693	103,672	5,234	108,906	21,593	263,286	8.2	0.925	14.0	12.5
1921	19,783,797	111,374	9,077	120,451	92,676	69,230	307	69,537	11,765	104,441	11.2	8.3	5.29	11.8
1922	35,602,826	101,612	95,007	256,619	202,119	70,252	8,151	78,403	12,878	214,497	5.8	8.3	6.003	13.5
1923*	44,943,696	251,988	113,833	365,771	287,910	129,683	23,340	153,023	25,078	312,688	8.0	8.2	0.690	15.6
1924*	37,931,939	211,965	56,588	268,553	211,046	107,067	37,165	144,232	25,164	236,210	10.7	7.8	0.600	13.9
1925*	45,893,839	260,149	77,697*	337,746	270,000*	65,747	†	65,747	13,150†	283,150	4.6	0.620	14.0	13.8
1926*	48,800,000	305,000	38,108*	343,108	274,486	73,500	†	73,500	14,700†	289,186	4.8	0.600	13.4	13.8
	583,149,441	3,131,757	1,088,281	4,220,038	3,255,391	1,850,680	105,293	1,955,973	380,949	3,036,040	10.55	0.618	13.9	

¹ International Control of Minerals. A. I. M. E. and M. & M. S. A. (1925) 55.² Figures 1922-26 supplied by author. Alloy tonnages 1923 and 1924 from Mineral Resources. (1924) Pt. 1, 100. Alloy tonnages 1925 and 1926 from *Iron Trade Review*. (Jan. 6, 1927) 86.

* All figures in gross tons.

† Includes spiegel.

‡ Assuming 80 per cent. alloy.

§ Included in ferro.

¶ Assuming 20 per cent. spiegel.

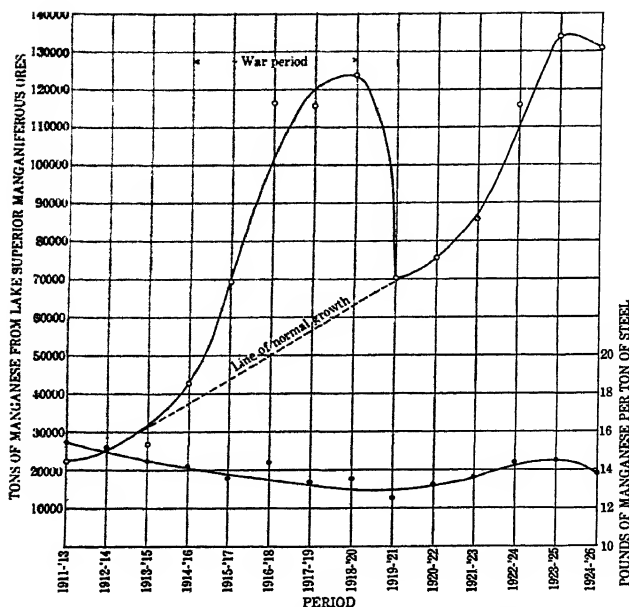


FIG. 1.—PRODUCTION OF MANGANESE FROM LOW-GRADE ORES AND QUANTITY USED IN STEEL BY 3-YR. PERIODS.

TABLE 2.—*Manganese from Low-grade Ores and Manganese Per Ton of Steel*

[Authority, Lake Superior Iron Ore Association]

Year	Ore, Tons	Manganese, Average Per Cent.	Manganese, Tons	Manganese, Lb. per Ton of Steel
1911	450,127	3.25	14,629	14.1
1912	827,154	3.02	24,980	14.2
1913	882,144	3.21	28,317	15.5
1914	622,290	3.63	22,589	15.4
1915	881,020	3.33	29,338	12.2
1916	1,794,325	4.34	77,874	14.8
1917	1,621,305	6.18	100,197	13.9
1918	2,613,135	6.61	172,728	14.5
1919	1,732,300	4.35	75,355	11.6
1920	2,377,560	5.23	124,346	14.0
1921	230,726	4.71	10,867	11.8
1922	1,592,266	5.88	93,625	13.5
1923	2,441,586	6.36	155,285	15.6
1924	1,438,241	7.00	100,677	13.9
1925	2,247,472	6.57	147,659	14.0
1926 ¹	2,248,500	6.57	147,500	13.4

¹ Approximate tonnages and analysis.

increase is somewhat surprising. The upper curve in Fig. 1 shows the increase in the consumption of manganese derived from Lake Superior manganiferous ores during the period 1911 to 1926. The peak in the curve is due to the fact that during the War considerable tonnages of manganiferous ores were used in the production of spiegel. In order to make a more direct comparison between the pounds of manganese per ton of steel and the amount derived from low-grade ores it would be desirable to know what portion of the manganese from manganiferous ores is used in making high-manganese basic pig iron. This information is not readily available. However, it is well known that the tendency in recent years has been toward basic iron containing more manganese, and it seems probable that most of the manganiferous ores are being used in basic pig-iron furnaces.

In 1925, 147,659 tons of manganese was shipped from the Lake Superior region, yet the consumption of manganese per ton of steel during this year was 14 lb., the average quantity for the 16-yr. period. During the last 6 years the amount of manganese obtained from manganiferous iron ores has apparently had little effect upon the amount of manganese required to make a ton of steel. It may be that the manganese specifications have been raised to such an extent as to more than offset the manganese saving effected by working to higher residual manganese made possible by using high-manganese pig iron in the basic open hearth. High-manganese, low-carbon steels are increasing in popularity and would tend to increase the manganese requirements. During the period when the amount of manganese per ton of steel was declining, prices were high. More recently prices have been easier and this is no doubt a factor which has a bearing upon the amount of manganese used to make a ton of pig iron. The practice of working to a high-residual manganese will be discussed in more detail later.

Since 1913 there has been a definite decrease in the ratio of pig iron to steel ingots. The practice of using more steel scrap together with the fact that there has been a tendency to substitute steel in the rolled or cast form for gray iron castings has lowered the ratio of pig iron to steel ingots from 101.2 in 1913 to 83.0 in 1926.² In other words, while the tendency has been to use higher-manganese iron, a smaller quantity of hot metal is now used in making up the basic open-hearth charge. The effect of this change which has taken place gradually is not commensurate with the increasing use of higher-manganese pig iron and does not offer an explanation as to why it is still necessary to use about 14 lb. of manganese in making a ton of steel in spite of the fact that more manganese is being charged into the basic open hearth with the hot metal.

² Steel-pig Iron Ratio Declining. (Editorial) *Iron Trade Rev.* (Jan. 20, 1927) 80, 209.

SUMMARY OF DOMESTIC MANGANESE REQUIREMENTS

The production of steel ingots and castings has increased almost 5-fold in the last 26 years. Although this tremendous growth has been irregular, with peak years followed by decided slumps in production, the general tendency has been toward a healthy growth. In considering future manganese requirements there seems to be no better guide for determining future steel production than past records. With this viewpoint Fig. 2 has been prepared showing the all-steel production by years from 1878 to 1926. Although it is admitted that the projection of

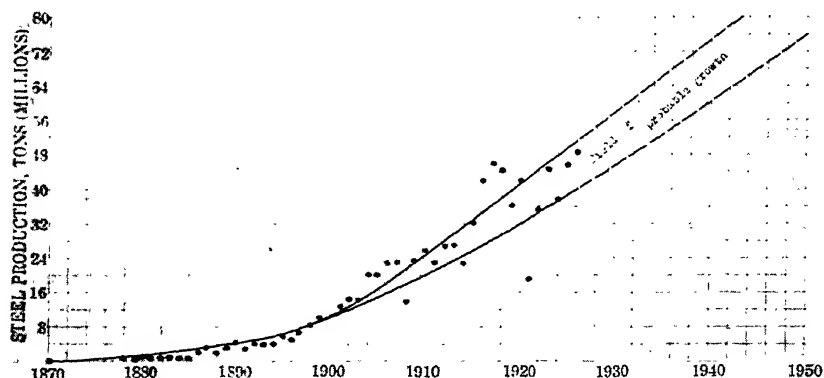


FIG. 2.—STEEL PRODUCTION OF UNITED STATES AND PROBABLE TREND.

the curve is uncertain, it seems probable that future growth will fall in the field so indicated in Fig. 2.

On the basis of the pounds of manganese required to make a ton of steel in the past 15 years and assuming the production of steel for the next 20 years will average 60,000,000 tons annually, the yearly requirement over the next 20 years may be summarized as follows:

	Tons
Assumed average yearly output of steel.....	60,000,000
Total metallic manganese required at 14 lb. per ton of steel.....	375,000
Tons manganese from spiegel, 10 per cent.....	37,500
Tons of manganese required as ferro.....	337,500
Tons of manganese from manganiferous ores.....	175,000

The approximate domestic manganese requirements can be converted into terms of domestic ores as follows:

Grade	Manganese Required, Tons	Conversion Loss, Per Cent.	Manganese Required in Ore, Tons	Manganese in Ore, Per Cent.	Ore, Tons
Ferro.....	337,500	20	421,875	42.2	1,000,000
Spiegel.....	37,500	25	50,000	16.0	400,000
Pig iron.....	122,500	30	175,000	8.8	2,000,000

DOMESTIC RESERVES

Relatively little attention was paid to the domestic manganese situation prior to the World War. Previous to that time the supply of manganese ores came largely from Russia and India. Imports from Brazil, Cuba and Central America assumed greater importance after 1914. During the stress of war the manganese situation became acute, inasmuch as this metal is practically indispensable in the manufacture of steel. Every effort was made to stimulate domestic production in view of threatening transportation difficulties. The War Industries Board which served during the World War and various Industrial Preparedness Committees which have served since the War have contributed much valuable information.

In considering the domestic reserves it is necessary to bear in mind that the major part of the manganese used in the steel industry is consumed in the form of ferromanganese. The ratio of manganese to iron largely governs the grade of alloy produced. For all practical considerations it may be assumed that all the iron and from 70 to 80 per cent. of the manganese will be recovered in the alloy. The ratio of manganese to iron in ferro is almost 7 to 1, making it obvious that the ore must show a higher ratio due to the loss of manganese which occurs in smelting. In the production of spiegeleisen lower grade ores may be used; the ratio of manganese to iron in this case stands approximately as 0.28 to 1. The recovery of manganese in spiegel practice is less than in ferro practice, averaging about 70 per cent. Another limitation is the phosphorus content of the ores. All the phosphorus will enter the metal, and inasmuch as more than two tons of ore will, in general, be required to make one ton of alloy, the phosphorus in the ore should be less than one-half the upper limit in the alloy. Figures for the allowable percentages of phosphorus in spiegel and ferro have been given under Metallurgical Uses of Manganese. The upper limit of phosphorus in these alloys depends somewhat upon their manganese content.

In studying the domestic reserves the authors have endeavored to investigate various sources of information. The Sub-Committee of the Mining and Metallurgical Society of America in submitting the report on this matter, already referred to, says:

In brief, the story of manganese clearly demonstrates that under natural conditions the United States has practically no commercial high-grade manganese ores; and our inquiry need go no further in this direction. There remains to be considered, however, how highly artificial conditions need to be in order to shift important quantities of manganese-bearing material across the border-line from waste to ore.

This has been the subject of very careful and thorough inquiry by your Committee. It will be unnecessary here to go into the multitudinous details which underlie the results which have been reached. Suffice it to say that 1850 manganese deposits, alleged deposits, and prospects have been carefully reviewed; fresh information has been gathered and important camps have been visited; exhaustive studies have been

TABLE 3.—*Indicated Total Reasonably Possible Reserves at \$50. Index Price Ferro Grade, 35 Per Cent. Mn*

State	Crude, Tons ¹		Concentrate, Tons		Total, Tons		Manganese, Tons	
Alabama.....	80,000	80,000	5,000	10,000	5,000	10,000	2,250	4,500
Arizona.....			15,000	20,000	75,000	100,000	33,800	45,000
Arkansas.....	180,000	210,000	125,000	165,000	285,000	375,000	135,500	174,300
Batesville.....	600	1,000			600	1,000	300	500
Western.....								
Total.....	160,000	211,000	125,000	165,000	285,000	376,000	132,800	174,800
California.....	100,000	180,000			100,000	160,000	40,000	64,000
Colorado.....	150,000	250,000			150,000	250,000	52,500	87,500
Georgia.....			85,000	200,000	85,000	200,000	35,700	84,000
Montana.....								
Butte—Crab.....	135,000	450,000			135,000	450,000	43,000	162,000
Butte—L.C.....			75,000	125,000	75,000	125,000	31,500	52,500
Phillipsburg.....	270,000	315,000	70,000	140,000	340,000	455,000	136,000	182,000
Other.....	400	400			400	400	150	150
Total.....	405,400	765,400	145,000	265,000	550,400	1,030,400	210,250	396,650
Nevada.....	20,000	30,000			20,000	30,000	8,000	12,000
New Mexico.....	13,500	20,000			13,500	20,000	5,400	8,000
Oregon.....	1,000	2,000			1,000	2,000	11,200	20,800
Tennessee.....	3,000	5,000	27,000	50,000	28,000	52,000	15,200	22,800
Utah.....	9,000	12,000	37,000	55,000	40,000	60,000	12,000	5,400
Virginia.....			300,000	1,000,000	300,000	1,000,000	126,000	420,000
Seven Others.....	8,000	10,000			8,000	10,000	3,300	4,200
Chemical.....	930,500	1,545,400	739,000	1,765,000	1,669,500	3,310,400	686,510	1,349,650
	270,000	315,000			270,000	315,000	103,000	126,000
	660,500	1,230,400	739,000	1,765,000	1,399,500	2,995,400	578,510	1,223,650

¹ Idaho, Maryland, New Jersey, North Carolina, Oklahoma, Texas, Wyoming.² Long tons.

TABLE 4.—*Indicated Total Reasonably Possible Reserves at Prices Corresponding to \$50. Index Price for Ferro-grade Ores, Spiegel and Manganese, Pig Grades, 5 to 35 Per Cent. Mn*

State	Crude, Tons ²	Concentrates, Tons	Total, Tons	Mn for Spiegel, Tons	Mn for Mn-pig, Tons
Arkansas.....	50,000	130,000	180,000	38,000	2,500
California.....	50,000		50,000	75,000	3,750
Colorado.....	1,500,000		1,500,000	2,500,000	
Georgia.....		115,000	115,000	375,000	625,000
Michigan.....	1,030,000		1,030,000	23,000	40,000
Minnesota.....	26,000,000		1,030,000	6,000	10,000
Montana.....			26,000,000	935,000	1,360,000
Philipsburg.....		50,000	50,000	15,000	22,500
Other.....	3,000		3,000	540	900
Nevada.....	200,000		200,000	300,000	36,000
New Jersey.....		5,000,000	5,000,000	600,000	720,000
New Mexico.....	500,000		500,000	80,000	160,000
Tennessee.....		5,000	5,000	10,000	2,500
Virginia.....	500,000	70,000	70,000	17,500	37,500
Wisconsin.....	4,000		500,000	1,000,000	25,000
Three Others ¹		1,000	5,000	1,000	1,500
	20,837,000	5,371,000	35,208,000	2,114,290	2,147,500
			48,118,000	3,064,700	2,858,750

¹ Idaho, Mississippi, Oklahoma.² Long tons.

made of past performance, with tabulations of geological features and associations in their relation to production; war records of production and cost and war claims for losses have been examined and tabulated in their bearing on the price necessary to bring out important quantities. In short, every available source of information has been made to contribute its quota, and your Committee feels justified in claiming due consideration for the results which will here be set forth in summarized form only.

Table 3, also from the Committee's report, shows the domestic reserves of ferro-grade ores that contain not less than 35 per cent. manganese. Attention is called to the fact that chemical grade ores fall in this class indicated by the manganese content, and have been set aside for this purpose. Fortunately, the situation with regard to low-grade ores (5 to 35 per cent. Mn) is not as serious as in the case of ferro ores. Table 4, also from the Committee's report, shows the total reserves of spiegel and manganese pig grades.

DOMESTIC RESERVES OF 5 TO 35 PER CENT. ORE

Minnesota Deposits

The relative importance of Minnesota deposits of low-grade ore compared to the total reserves is shown in Table 4. Minnesota deposits have been estimated to contain 26,000,000 to 35,000,000 tons of 5 to 35 per cent. manganese ore, while the total domestic deposits of such ore have been estimated to contain from 35,000,000 to 48,000,000 tons. Using the mean of these estimated tonnages, it is evident that about 75 per cent. of the reserves are in Minnesota.

Brown and Black Ores of Cuyuna District

There are two general types of ores which have been designated by Newton³ as low-phosphorus, high-silica ore, and high-phosphorus ore. Zapffe⁴ refers to the same types of ores and has called the former black ores and the latter brown ores. Table 5 gives season analyses typifying various black ores and Table 6 gives typical season analyses of various brown ores.

It can be seen by glancing at Tables 5 and 6 that (1) the black ores contain from 10 to 15 per cent. less iron than the brown ores, (2) the black ores are considerably lower in phosphorus but higher in manganese, and (3) the moisture content of the brown ores varies over a wide range and, in general, is higher than in the black ores.

³ Edmund Newton: Manganiferous Iron Ores of the Cuyuna District, Minnesota. *Bull.* 5, Univ. Minn. School Mines Exper. Sta. (1918) 10.

⁴ Carl Zapffe: Manganiferous Iron Ores of the Cuyuna District, Minnesota. *Trans.* (1925) 71, 372-85.

TABLE 5.—*Season Analyses Typifying Various Black Ores, According to Zapffe*

Ore	Fe (Dry), Per Cent.	P, Per Cent.	SiO ₂ , Per Cent.	Mn, Per Cent.	Al ₂ O ₃ , Per Cent.	Mois- ture, Per Cent.
Algoma.....	32.00	0.07 to 0.09	20 to 23	17 to 19	2 to 3	7 to 9
Clarke.....	35.50	0.104	15.28	16.95		
Clarke.....	35.00	0.058	23.86	10.99		
Cuyuna-Mille Lacs.....	37.00	0.09 to 0.10	18 to 20	11 to 12	13 to 14
Cuyuna-Mille Lacs.....	36.73	0.106	12.93	17.44	13.48
Cuyuna-Mille Lacs.....	37.46	0.106	7.97	22.01	13.50
Ferro.....	29 to 32	0.07 to 0.09	17 to 19	20 to 21	7 to 10
Gloria.....	28.00	0.180	17 to 19	17 to 18	8 to 9
Hopkins (black ore, expected analysis in 1919).....	38.00	0.063	16.45	16.03	1.73	9.91
Mangan No. 1, crude.....	36.00	0.130	22.00	13.00	2.50	9 to 10
Mangan No. 1, hand picked..	33.00	0.130	17.00	17.00	2.50	9 to 10

TABLE 6.—*Season Analyses Typifying Various Brown Ores, According to Zapffe*

Ore	Fe (Dry), Per Cent.	P, Per Cent.	SiO ₂ , Per Cent.	Mn, Per Cent.	Al ₂ O ₃ , Per Cent.	Moisture, Per Cent.
Arko.....	41	0.290	9	10 to 12	4	14 to 15
Armour, No. 2....	46 to 49	0.18 to 0.21	8.5 to 9.5	5.2 to 8.2	3.5 to 4.5	11 to 13
Hillcrest.....	46 to 49	0.25 to 0.28	6 to 9	4 to 6	3.8 to 4	8.5 to 9.5
Huntington.....	53.65	0.309	7.29	2.78	2.92	9.56
Milford.....	43.05	0.238	6.70	11.41	4.00	18.00
Martin.....	52.81	0.182	7.60	5.04	13.50
Mahnomen.....	50.14	0.277	6.97	4.93	3.24	12.00 ¹
Mahnomen.....	46.80	0.283	5.48	8.25	4.20	12.00 ¹
Mahnomen.....	43.33	0.297	5.10	11.72	3.25	12.00 ¹
Mahnomen.....	41.03	0.296	4.43	13.89	3.25	12.00 ¹
Mahnomen.....	36.42	0.310	3.90	18.69	3.20	12.00 ¹
Sagamore.....	43.00	0.280	5.00	10.00	5.00	10.00 ²
Sagamore.....	48.00	0.280	8.00	6.50	4.00	10.00 ²
Sagamore.....	51.00	0.280	8.00	5.00	4.00	10.00 ²
Sagamore.....	53.50	0.280	8.00	3.50	3.50	10.00 ²
Sultana.....	38.00	0.174	9 to 10	13.75	3.50	15.00
Portsmouth.....	46.20	0.286	6.44	6.32	4.52	17.00
Portsmouth.....	43.22	0.298	6.33	9.55	4.84	17.00
Armour No. 2....	46.88	0.201	10.43	7.21	4.07	11.31
Louise.....	45.95	0.213	8.72	8.98	2.98	14.00

¹ Washed ores, 1923.² Dried ores, expected analyses for 1924.

Size of Deposits

In considering the information relative to the tonnages of ore on the Cuyuna range, it should be borne in mind that the first shipments⁵ were made in 1913. At that time 24,434 tons were shipped for use experimentally in small lots. Earlier development was undoubtedly retarded due to the entrance of the U. S. Steel Corp. into the district in 1905,⁶ and later abandonment of options after considerable exploration work. Estimated tonnages based upon drilling and whatever other data are available are usually a good guide to determine whether a deposit should be developed, but it should be remembered that the error in such figures may be large, particularly in the case of irregular deposits. As the range develops, more information will become available as to the amounts and composition of deposits.

Distribution of Brown and Black Ores According to Composition

The following tables,⁷ based upon drilling records and other available information, represent estimates of manganiferous iron orebodies, segregated according to tonnage and composition.

MANGANIFEROUS CONTENT OF BLACK ORES

Group 1

Tons	Iron, Per Cent.	Phosphorus, under 0.1 Per Cent.	Silica, under 20 Per Cent.	Manganese, 10 to 15 Per Cent.
437,235	42.32	0.079	13.00	14.58
1,446,080	41.68	0.069	12.75	12.80
1,883,315	41.83	0.071	12.81	13.21

Group 2-A

Tons	Iron, Per Cent.	Phosphorus, under 0.1 Per Cent.	Silica, under 20 Per Cent.	Manganese, above 15 Per Cent.
57,964	30.62	0.095	18.74	19.37
28,000	30.57	0.093	19.40	19.19
84,316	30.11	0.077	18.13	19.95
125,938	34.47	0.089	12.68	19.72
275,412	36.09	0.082	13.85	18.13
160,833	20.09	0.080		18.69
732,463	30.96	0.084	14.99	18.87

⁵ Carl Zapffe: *Op. cit.*, 375.⁶ Edmund Newton: *Op. cit.*, 3.⁷ E. M. Lambert: Minn. School Mines; personal communication.

Group 2-B

Tons	Iron, Per Cent.	Phosphorus, 0.1 to 0.2 Per Cent.	Silica, under 20 Per Cent.	Manganese, above 15 Per Cent.
52,500 ¹	35.00	0.155		17.83
36,718	34.31	0.182	16.53	15.32
35,312	28.95	0.138	17.77	17.91
176,644	31.11	0.128	15.68	18.74
46,760	29.67	0.122	14.06	25.20
76,666	32.95	0.198	12.18	19.23
<hr/> 424,600	<hr/> 31.86	<hr/> 0.149	<hr/> 15.04	<hr/> 19.06

¹ Non-merchantable tonnage.

Group 3

Tons	Iron, Per Cent.	Phosphorus, Per Cent.	Silica, above 20 Per Cent.	Manganese, Per Cent.
86,120	30.94	0.147	21.47	17.86
83,546	28.83	0.104	20.75	17.79
5,190	30.00	0.150	20.00	17.00
1,099	29.81	0.070	24.99	16.79
186,873	30.21	0.148	23.00	15.56
<hr/> 362,828	<hr/> 30.10	<hr/> 0.138	<hr/> 22.10	<hr/> 16.65

Total Tonnage Black Ores

Tons	Iron, Per Cent.	Phosphorus, Per Cent.	Silica, Per Cent.	Manganese, Per Cent.
3,403,206	36.99	0.091	14.55	15.53

Average size of deposit 179,116 tons (black ores).

MANGANIFEROUS CONTENT OF BROWN ORES

Group 1-A

Tons	Iron, Per Cent.	Phosphorus, under 0.2 Per Cent.	Silica, Per Cent.	Manganese, Up to 5 Per Cent.
36,153 ¹	50.52	0.144	6.12	4.73

¹ Non-merchantable tonnage.

Group 1-B

Tons	Iron, Per Cent.	Phosphorus, 0.2 to 0.3 Per Cent.	Silica, Per Cent.	Manganese, Up to 5 Per Cent.
2,731,820 ¹	52.50	0.299	7.37	2.62
1,393,202 ²	54.16	0.220	8.58	1.94
54,959	48.53			4.72
373,652	53.03	0.285	9.80	3.60
4,553,633	42.02	0.273	7.95	2.52

¹ About 25 per cent. manganiferous ore and 75 per cent. non-Bessemer ore.² About 40 per cent. manganiferous ore and 60 per cent. non-Bessemer ore.

Group 2-A

Tons	Iron, Per Cent.	Phosphorus, under 0.1 Per Cent.	Silica, Per Cent.	Manganese, 5 to 10 Per Cent.
146,100	54.75	0.036	3.87	8.33

Group 2-B

Tons	Iron, Per Cent.	Phosphorus, under 0.2 Per Cent.	Silica, Per Cent.	Manganese, 5 to 10 Per Cent.
71,314	49.38	0.151	3.79	8.91
903,993	49.90			6.42
185,000 ¹	45.39			5.85
460,937	47.27	0.154		8.50
33,750	42.12	0.197		8.42
1,654,994	48.48	0.156	3.79	7.08

¹ Non-merchantable tonnage.

Group 2-C

Tons	Iron, Per Cent.	Phosphorus, 0.2 to 0.3 Per Cent.	Silica, Per Cent.	Manganese, 5 to 10 Per Cent.
5,586	49.60	0.290	5.17	6.35
372,104	49.60	0.290	5.17	6.35
635,912	47.00	0.205	10.50	6.00
57,847	46.18	0.234	7.01	8.73
10,836,026	43.15	0.291	5.00	9.89
24,000	45.29	0.259		6.75
75,000	47.50	0.212		6.11
12,006,475	43.60	0.285	5.31	9.54

Group 2-D

Tons	Iron, Per Cent.	Phosphorus, over 0.3 Per Cent.	Silica, Per Cent.	Manganese, 5 to 10 Per Cent.
748,925	49.53	0.484		5.74
689,326	45.76	0.320	6.16	8.37
680,949	44.57	0.334	7.05	9.60
191,482	47.52	0.336	6.43	8.00
<u>2,310,682</u>	<u>46.78</u>	<u>0.378</u>	<u>6.58</u>	<u>7.85</u>

Group 3-A

Tons	Iron, Per Cent.	Phosphorus, 0.1 to 0.2 Per Cent.	Silica, Per Cent.	Manganese, 10 to 15 Per Cent.
539,931	41.83	0.107	11.86	10.89
204,107	41.21	0.185	9.00	10.83
427,811	44.28	0.182		10.50
13,750 ¹	42.33			11.38
56,562	33.12	0.166	14.59	14.22
154,870	39.18	0.167	11.65	10.01
<u>1,397,031</u>	<u>41.85</u>	<u>0.151</u>	<u>7.86</u>	<u>10.80</u>

¹ Non-merchantable tonnage.

Group 3-B

Tons	Iron, Per Cent.	Phosphorus, 0.2 to 0.3 Per Cent.	Silica, Per Cent.	Manganese, 10 to 15 Per Cent.
937,619	42.85	0.237	6.32	11.26
33,356	43.00	0.220	8.00	11.00
578,753	39.62	0.270	11.00	11.60
11,700	39.28	0.200		13.10
696,299	40.54	0.260	6.18	10.55
<u>2,257,727</u>	<u>41.29</u>	<u>0.252</u>	<u>7.51</u>	<u>11.13</u>

Group 3-C

Tons	Iron, Per Cent.	Phosphorus, over 0.3 Per Cent.	Silica, Per Cent.	Manganese, 10 to 15 Per Cent.
36,811	36.32	0.335	21.00	13.69
186,802 ¹	42.84	0.353		10.76
<u>223,613</u>	<u>41.77</u>	<u>0.350</u>	<u>21.00</u>	<u>11.24</u>

¹ Non-merchantable tonnage.

Group 4

Tons	Iron, Per Cent.	Phosphorus, 0.2 to 0.3 Per Cent.	Silica, Per Cent.	Manganese, above 15 Per Cent.
328,500	38.94	0.206	9.55	15.14

TOTAL TONNAGE AND AVERAGE COMPOSITION OF BROWN ORES CONTAINING MORE
THAN 5 PER CENT. MANGANESE

Tons	Iron, Per Cent.	Phosphorus, Per Cent.	Silica, Per Cent.	Manganese, Per Cent.
20,325,122 ¹	43.96	0.27	6.00	9.50

¹ Total tonnage exclusive of groups 1-A and 1-B.

Average size of brown ore deposit 655,649 tons, exclusive of groups 1-A and 1-B.

The grouping followed in the foregoing tables was adopted arbitrarily to show the composition and relative size of known orebodies. It is readily seen by glancing at the total tonnages for the various groups of black ores that the low-phosphorus material predominates. Groups 1 and 2-A, averaging 0.071 and 0.084 per cent. phosphorus, respectively, contain about 75 per cent. of the black ores. Group 2-B, in the same range of manganese as group 2-A but with higher phosphorus, is little more than half as large. Group 3 includes deposits of black ore which contain more than 20 per cent. silica. A discussion of deposits which have not been classed as ore and which would fall in this group will be given later.

Groups 1-A and 1-B of the brown ores have not been included in the total tonnage of this type of ore because the material in these groups does not fall in the class of manganiferous iron ores according to the most recent classification adopted by the U. S. Geological Survey.

In 1916 Hewett⁸ classified manganese ore as follows: manganiferous iron ore, 5 to 15 per cent. manganese; ferruginous manganese, 15 to 40 per cent. manganese; manganese ore, more than 40 per cent. manganese. Jenison⁹ has modified this classification, including as manganiferous iron ore materials containing from 5 to 10 per cent. manganese in place of 5 to 15 per cent. The ferruginous manganese ores were changed accordingly to include material containing from 10 to 35 per cent. manganese. Material containing more than 35 per cent. manganese is considered as manganese ore. The grade of manganese ore was lowered from 40 to 35 per cent. during the World War in keeping with the change from 80 to 70 per cent. ferro.

It can readily be seen that the brown ores are higher in phosphorus and lower in silica than the black ores. Group 2-C in the brown ores

⁸ D. F. Hewett: Mineral Resources, Part 1. U. S. Geol. Surv. (1916) 733-34.

⁹ H. A. C. Jenison: Mineral Resources, Part 1. U. S. Geol. Surv. (1919) 98-101.

contains about 60 per cent. of the total brown ore tonnage. The bulk of the brown ore runs from 0.2 to 0.3 per cent. phosphorus. The total tonnage of brown ore, according to foregoing figures, is between six and seven times as great as the total tonnage of black ore. The average size of deposits is considerably larger in case of the brown ore. In groups 1-A and 1-B deposits in the same range of manganese were segregated according to phosphorus content. The same scheme was followed in groups 2-A, 2-B, 2-C and 2-D; all deposits containing from 5 to 10 per cent. manganese being grouped according to the percentage of phosphorus they contain. Groups 3-A, 3-B and 3-C follow the same scheme in the range of 10 to 15 per cent. manganese. These groups, strictly speaking, should be called ferruginous manganese ores. Group 4 contains one deposit of more than 15 per cent. manganese.

COMPARISON OF DOMESTIC REQUIREMENTS AND RESERVES
[Estimated Average Annual Manganese Requirements for next 20 Years]

Grade	Manganese Required, Tons	Conversion Loss, Per Cent.	Manganese Required in Ore, Tons	Manganese in Ore, Per Cent.	Ore, Tons
Ferro.....	337,500	20	421,875	42.2	1,000,000
Spiegel.....	37,500	25	50,000	16.0	400,000
Pig iron.....	122,500	30	175,000	8.8	2,000,000

The domestic reserves have been summarized¹⁰ as follows:

Grade	Estimated Reserves of Ore, Tons	Manganese, Average Per Cent.	Manganese	
			Contained, Tons	Recoverable, Tons
Ferro.....	1,400,000	41.3	578,000	433,000
Spiegel.....	13,168,000	16.00	2,114,290	1,480,000
Manganese pig.....	22,050,000	9.75	2,147,500	1,500,000

The foregoing figures indicate that our reserves of ferro-grade ores are totally inadequate and that although our reserves of spiegel and high-manganese pig grades are more adequate, every effort should be made to conserve them. The outlook with regard to spiegel and high-manganese pig is not nearly as serious as in the case of ferro-grade ores because these demands can be met with lower grade material if necessary. However, it seems certain that if present tendencies continue the best of the low-grade material will be exhausted within the next 20 years.

The situation in brief is that we have very limited reserves of ferro-grade ores and our low-grade material which might be used in case of

¹⁰ International Control of Minerals. A. I. M. E. and M. & M. S. A. (1925) 83.

necessity to produce ferro or intermediate alloys is being used at a rapid rate. There are sufficient tonnages of manganiferous ores in Minnesota to supply the needs for high-manganese pig iron for 20 to 30 years, assuming that the steel industry will make whatever adjustments are necessary to meet some decline in quality toward the end of such a period.

ULTIMATE POSSIBILITIES OF THE CUYUNA RANGE

It is impossible to predict with any degree of accuracy the ultimate tonnage of manganiferous iron ore which will be produced on the Cuyuna range. Zapffe¹¹ states that the brown ore deposits are in a stage of development suitable for accurate estimating. He also states, however, that the reserves of manganiferous iron ores are not as well known as they should be. The authors believe that the figures given in the foregoing tables are conservative and that tonnages much larger than those given will ultimately be produced.

Group 3 of the black ores includes deposits containing more than 20 per cent. silica, which have been regarded as merchantable ore. There are, however, large tonnages of high-silica manganiferous ore which are not merchantable today but should nevertheless be regarded as probable future ore. Zapffe,¹² in referring to this class of material, states:

Because the black ores are usually associated with large widths of siliceous manganiferous formation, conditions are favorable for the development of large open pits on it and enable low mining costs to be obtained. The ore produced, if we may call it that, will be higher in manganese than in the present brown ore. Also, that material will be low in phosphorus, alumina and moisture. In silica it will range between 25 and 35 per cent. Iron and manganese combined will total about 40 to 45 per cent. dry. In one place alone there is enough such material in a continuous band, for a mile or more, to produce a tonnage far in excess of any maximum figures now producible for black and brown ore reserves combined. There are other big areas. To some people it may seem absurd to refer to or consider material so high in silica, but it is the use of manganese and growing practice that are under consideration and 20 years hence something must be offered to replace the rapidly depleted brown ores. Due to low-cost mining and the higher manganese content this material will not be shunned. It will offer new problems, but they will be met. This material will have an advantage in contributing more manganese per ton than do the brown ores now being taken eagerly.

Van Evera¹³ believes that 20,000,000 tons might be added to the estimated tonnages given for black as well as brown ores. The tonnage in the case of the black ores includes high-silica material which at present must be classed as probable future ore.

¹¹ Carl Zapffe: *Op. cit.*, 379-80.

¹² *Ibid.*, 381-82.

¹³ Wilbur Van Evera: Written communication with T. L. Joseph.

Crosby¹⁴ thinks that the possibilities are very promising for developing the various types of manganiferous ore on the Cuyuna range in quantities greater than those given in the foregoing tables.

Bradley¹⁵ believes that the ultimate tonnage of black ores containing more than 20 per cent. silica will greatly exceed the figures given in Group 3 of this type of ore.

In general the black material which has been regarded as ore occurs in narrow bands and small tonnages are the rule. Underground mining methods would characterize extraction of these ores. Most of the mines which produced during the World War are now idle, not because the deposits have been exhausted but due to a lack of demand for these ores for the production of high-manganese pig iron. Low-phosphorus ores are not essential in basic iron as they are in producing spiegeleisen. The high silica content of the black ores increases the slag volume, which in turn increases the fuel consumption of pig iron furnaces. However, inasmuch as their manganese content is in general higher than that of the brown ores, smaller quantities would be necessary in order to get 2 per cent. manganese in the metal. Other factors, such as size of deposits, availability, and uniformity of composition, have probably been more important in developing the brown ores to the exclusion of the black ores.

When a program looking toward the utilization of these ores was first considered, the question immediately arose as to what type of ore should be investigated. The purpose of the investigation was to develop more efficient ways of utilizing a limited resource and to determine the possibility of using manganiferous iron ores in the production of ferromanganese. With this general viewpoint the desirability of developing methods which can be applied to the ores which occur most abundantly is obvious. A study of the best information available plainly indicates that one problem lies in developing a process for the treatment of the brown ores which contain from 8 to 12 per cent. manganese and 0.2 to 0.3 per cent. phosphorus.

The first efforts have been directed toward determining the possibilities of making ferro from low-grade ores by using present iron and steel furnaces, perhaps with slight modifications. Experimental work thus far has been confined largely to the brown ores which, because of their higher iron content and relatively low percentage of silica, present a somewhat different problem than do the black ores. The two ores differ so widely in composition that it does not follow that the same process would apply to both types of ores. As the experimental program which is being followed for treating the brown ores approaches completion, attention will be turned toward the black ores.

¹⁴ George H. Crosby: Written communication with T. L. Joseph.

¹⁵ Wilson Bradley: Oral communication with T. L. Joseph.

BENEFICIATION OF CUYUNA RANGE ORES

Separation of the silica which ranges from about 10 to 20 per cent. in the black ores would be very desirable. Although the brown ores contain less gangue, their concentration would also be desirable. A thorough study has been made by the Minnesota School of Mines Experiment Station of the application of standard mechanical processes to both types of manganiferous iron ores. The conclusions drawn from this work are as follows:¹⁶

Tests previously given are representative of many others that have been made. The samples tested do not represent all types of material which may be encountered in the district. The results of tests, however, bear out the previously stated opinions gained from a careful study of the physical character of the ore and of an interpretation of the method by which it was formed. As the result of study and experimental data, it is apparent that the beneficiation of Cuyuna manganiferous iron ores by means of present standard mechanical processes can accomplish but little. From an economic point of view the improvement in grade results in little if any increased profits.

Additional experimental work based upon a more complicated method of treatment, such as leaching, may eventually evolve a satisfactory technical method of treatment. Whether such process would be applicable only in times of emergency such as at present has yet to be determined. Therefore it is well to realize the difficulties of improving the character of the material and endeavor to utilize various ores in their natural condition by means of slightly modified metallurgical practice.

It should be mentioned in passing that the foregoing investigation included tests on magnetic separation, following a reducing roast designed to convert the iron into magnetite. Davis¹⁷ has made similar tests on Cuyuna ores and reports unfavorable results.

That the Cuyuna range manganiferous ores are not readily amenable to usual methods of beneficiation is borne out by the following microscopic examination of samples of black and brown ore. Data resulting from an examination¹⁸ of the brown ore follow:

1. The samples of Cuyuna iron-manganese ore examined contained hematite, limonite, psilomelane, a calcium phosphate mineral which is probably apatite or one of its modifications, and a small amount of siliceous gangue.
2. Psilomelane occurs as veins or lenses and in interbedded stratified form intimately associated with the iron oxides.
3. Although psilomelane was the only manganese mineral identified, manganese in other forms probably occurs in this type of ore.

¹⁶ Edmund Newton: *Op. cit.*, 57.

¹⁷ E. W. Davis: Magnetic Concentration of Iron Ore. *Bull.* 9, Univ. Minn. School Mines Exper. Sta. (1921) 115.

¹⁸ Examination by R. E. Head, U. S. Bur. Mines, Salt Lake City, Utah.

4. As a rule both the gangue minerals and the oxide manganese occur in a sufficiently fine state of division in the ores to require relatively fine grinding to effect their liberation.

5. The identification of the calcium-phosphate mineral as apatite or one of its modifications is based on optical data and qualitative microchemical tests. In any event its separation from the balance of the ore would be attended with the same difficulties as the elimination of the gangue and the segregation of the manganese and iron oxides into separate products.



FIG. 3.—PHOTOMICROGRAPH OF CUYUNA ORE, SHOWING INTIMATE ASSOCIATION OF IRON AND MANGANESE OXIDES. BLACK AREAS REPRESENT MANGANESE.

6. A general idea of the mode of association of the manganese and iron oxides and the form in which the phosphorus occurs in the brown Cuyuna ore may be gained from Figs. 3 and 4.

Thin sections of a sample of black ore showed that the distribution of the gangue was fairly uniform and consisted chiefly of quartz grains. The greater portion of these quartz grains are of very small size and a goodly number approach 0.05 mm. dia. (300-mesh screen opening equals 0.043). The average maximum diameter of the quartz grains is 0.10 mm. (150-mesh opening equals 0.104 mm.).

It is evident therefore that any attempt to beneficiate this type of ore would involve fine grinding in order to unlock the quartz particles from the associated manganese and iron oxides. Since finely

pulverized manganese oxide would probably have a tendency to adhere to the surfaces of the gangue particles, it appears doubtful whether clean quartz surfaces would be produced during crushing. Microscopic examination of crushed material indicates that this difficulty would be encountered.

In considering the ore as a whole it is evident that its beneficiation presents a difficult problem, due largely to the physical characteristics as noted above.



FIG. 4.—SHOWS THE CALCIUM PHOSPHATE ENCRUSTED ON THE WALLS OF A VUG IN THE REDDISH-BROWN CUYUNA ORE.

PRESENT USE OF MANGANIFEROUS IRON ORES

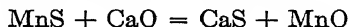
Before taking up new methods for treating manganiferous iron ores it seems advisable to discuss briefly the present use of these ores. The demand for manganese ores was greatly stimulated during the World War when imports of manganese ores and alloys were restricted. At the close of the War when low-phosphorus ores were no longer in demand, attention turned to the brown ores which occur in larger deposits, are more uniform in composition, and can in general be mined more cheaply. The phosphorus which is particularly objectionable in ores used in making spiegel is not prohibitive for making high-manganese pig iron inasmuch as this impurity can be removed in the basic open-hearth furnace.

MANGANESE AND BLAST-FURNACE PRACTICE

Under existing practice sufficient manganese iron ores are added to the blast-furnace charge to produce pig iron to contain approximately 2 per cent. manganese. In the smelting process about 75 per cent. of the manganese is recovered in the metal and the major portion of the remaining 25 per cent. passes into the slag. The presence of manganese in the charge aids desulfurization of the iron in the lower part of the furnace. It is well known that manganese will desulfurize iron according to the following reaction:



Manganese sulfide is less soluble in molten iron than iron sulfide. As a result the manganese sulfide rises from the metal, enters the slag, and eliminates sulfur as manganese sulfide. A portion of the sulfur is probably converted into calcium sulfide as follows:



Ferromanganese, spiegeleisen, and pig iron containing from 6 to 10 per cent. manganese contain only traces of sulfur. There is little doubt that manganese aids desulfurization. The extent of desulfurization depends upon the amount of manganese present. Although it is impossible to state with mathematical definiteness the desulfurization accomplished by manganese in blast furnaces making 2 per cent. pig iron, it is believed to be of an order of magnitude which makes it of practical importance.

Another benefit from manganese seldom referred to is its effect upon the free-running of the slag. Manganese in general makes a more fluid slag and one with a lower free-running temperature. The lower the free-running temperature of the slag the greater will be the difference between tuyere temperatures and slag temperatures. If a slag can be produced which will have a low free-running temperature, additional burden can be put on the furnace up to a point where the temperature in the combustion zone is still high enough to render possible the necessary hearth reactions such as desulfurization and reduction of metalloids. Slags which have high free-running temperatures do not pass quickly through the bosh and combustion zone. The ash liberated from combustion is not as readily absorbed by such slags; as a result the coke does not present a carbon surface for combustion. Furnace-men in general prefer to work with slags which are slightly acid because such slags usually have lower free-running temperatures than more basic slags.

The solid charge in the blast furnace must descend against the pressure exerted by the ascending gases. Any marked tendency to increase the pressure acting upward on the charge column is likely to interfere with stock descent and result in hanging and slipping which are undesirable. As the reduced ore (85 per cent.) approaches a position in the furnace somewhere in the bosh, fusion of the ore gangue and flux takes place.

The zone is not necessarily sharp, but there is a pasty zone which retards the ascent of the gases, resulting in increased pressures. Manganese acts to increase fluidity and lowers the melting points of slags. Below the fusion zone in the bosh and combustion zone there is considerable slag passing down over the coke; greater fluidity decreases the pressure here as well as in the zone of fusion.

DESIRABILITY OF MANGANESE IN BASIC IRON

Sulfur control belongs essentially to the blast furnace and can not be carried out economically in the open hearth. It is possible to eliminate sulfur in the open hearth by use of additional limestone or ferromanganese, but both methods are costly and uncertain. Elimination of sulfur in the blast furnace is more certain than in the open hearth, but it makes the operation more difficult to control and less economical. As low-sulfur coke becomes increasingly scarce, sulfur will become more and more of a problem. It has been mentioned that manganese aids desulfurization in the blast furnace. During the time the metal is in the transfer ladles between the blast furnace and the mixer and on its way to the open-hearth furnace from the mixer, manganese eliminates sulfur. Herty and Gaines¹⁹ have summarized their data on the desulfurizing action of manganese in iron as follows:

Within the range covered by these data the following conclusions are drawn:

1. In ladles of molten pig iron the equilibrium between manganese and sulfur at 2400° F. is expressed by the relation

$$(\text{per cent. Mn}) (\text{per cent. S}) = 0.070,$$

when the product (per cent. Mn) (per cent. S) is above 0.070 at the blast furnace, and when no blast-furnace slag is present on the iron.

2. If the product (per cent. Mn) (per cent. S) is above 0.070 at the blast furnace elimination of sulfur will take place until equilibrium is established.

3. If the product (per cent. Mn) (per cent. S) is less than 0.070 at the blast furnace little or no elimination of sulfur will take place.

4. If blast-furnace slag is present in the ladle, sulfur may be reduced from the slag into the metal.

5. If the manganese sulfide eliminated from the iron is poured into the open hearth the advantage of desulfurization by high manganese is lost.

Wheaton²⁰ has sampled 136 ladles of iron and grouped the samples according to manganese content; first group, 1 to 1.5 per cent. Mn;

¹⁹ C. H. Herty, Jr., and J. M. Gaines, Jr.: Desulfurizing Action of Manganese in Iron. See p. 434.

²⁰ E. A. Wheaton: The Use of High Manganese Iron. Yearbook, Amer. Iron Steel Inst. (1920) 399.

second group, 1.5 to 2.0 per cent. Mn; third group, 2 per cent. and above. Average results for the three groups follow:

	Manganese at Blast Furnace, Per Cent.	Sulfur at Blast Furnace, Per Cent.	Sulfur at Open-hearth Mixer, Per Cent.
Average, 29 tests.....	1.39	0.0867	0.041
Average, 89 tests.....	1.72	0.070	0.035
Average, 26 tests.....	2.13	0.073	0.031

These data confirm the statement of Kinney²¹ who says:

. . . 0.04 per cent. sulfur steel can be regularly and economically produced from such iron (2 per cent. Mn) even though its sulfur content rises to 0.07 per cent.

Smith²² lists the advantages and disadvantages of using manganiferous pig iron in the open hearth as follows:

ADVANTAGES

1. Elimination of sulfur.
2. Purifying action in the bath, which tends to produce a better quality steel.
3. Saving ferromanganese.
4. Decreased lime charge and the saving of fluorspar.
5. Increased tonnage.

DISADVANTAGES

1. The disadvantages in the blast-furnace operation.
2. The metallic loss in the open-hearth practice.
3. The increased corroding action on open-hearth basic refractories and ladle refractories.

He also summarizes the advantages and disadvantages of using manganiferous ores in the blast furnace as follows:

ADVANTAGES

1. The higher manganese tends to increase the rate of driving, which is due to a more fluid iron.
2. Troubles due to high bottoms are reduced.
3. Less runner and ladle scrap is produced in the manufacture of high-manganese iron than with low-manganese iron.
4. As a general rule, higher blast temperature can be used if the manganese of the pig is not carried too high. Lower coke consumption is obtained than in the production of lower manganese pig iron.

²¹ C. L. Kinney, Jr.: Economic Significance of Metalloids in Basic Pig Iron in Basic Open-hearth Practice. *Trans.* (1924) **70**, 165.

²² A. W. Smith: The Use of High-manganese Basic Pig Iron and Manganiferous Ores in Open-hearth Steel Practice. Paper before Amer. Iron Steel Inst. (1926).

5. With better furnace operations less flue dust is produced.

6. Owing to the affinity of manganese for sulfur, the use of high-manganese iron will enable the open hearth to produce steel of lower sulfur content.

DISADVANTAGES

1. The irregular manganese content in the ore over a period of one ore season, which as a rule results in an irregular content of manganese in the pig iron produced, inasmuch as ordinary iron ores are not classified on a manganese basis.

2. If pig iron is produced with excessive manganese content, the furnace bottom is liable to become low and frequent break-outs may occur, due to the extremely bad cutting action on the brick-work of the furnace.

In the open-hearth process itself, manganese is largely oxidized. At the end of the heat a portion of the original manganese is still retained in the steel. This is known as residual manganese and varies in amount depending upon the manganese in the pig iron, the proportion of scrap used, whether the practice of a run-off slag is followed, the iron oxide content of the slag, the basicity of the slag, and temperature. Residual manganese decreases the quantity of ferromanganese necessary to reach a given manganese percentage in the finished steel. An added advantage accrues from the improvement in the quality of the steel. This benefit is not reflected on the open-hearth cost sheet but appears on the cost sheets of the finishing mills. Manganese imparts fluidity to open-hearth slags just as it does to blast-furnace slags. Although there is at present no way definitely to determine the importance of this effect on open-hearth practice, the majority of operators attribute definite benefits to a fluid slag.

Iron will absorb sulfur when heated in an atmosphere containing SO_2 . During the melting-down stage in the open hearth sulfur is absorbed in quantities which depend upon the amount of SO_2 in the gases. This is governed by the sulfur in the coal and its heating value in producer-gas practice. Manganese in the open-hearth charge aids in removal of sulfur which is imparted by the fuel. With a growing scarcity of low-sulfur producer fuel, the aid of manganese in desulfurization will become more and more important. In the case of low-manganese iron the sulfur in the pig iron should be almost as low as that specified in the steel. High-manganese pig will not solve the problem of high-sulfur coal but will, nevertheless, mitigate conditions.

RESIDUAL MANGANESE

In the open hearth the greater portion of the manganese in the pig iron enters the slag. The amount retained in the metal at the end of the heat, and commonly referred to as residual manganese, depends upon

plant practice. Kinney²³ gives 8 types of open-hearth heats, and reports residual manganese from 0.12 to 0.40. The latter value was attained under most favorable conditions, *i. e.*, high-manganese iron, low-silica ore charge and no run-off slag. Pazetti²⁴ reports a month's practice using 1.65 manganese in the iron and shows a residual manganese of 0.22 when using no run-off slag. He says further:

In a plant using a run-off slag, the record of five carefully observed heats showed average residual manganese to be 0.09 per cent. when average manganese in pig iron was 1.69 per cent.

It is recognized that commercial expediency determines very largely the practice which will prevail in individual plants. In some plants the use of high-manganese pig iron saves from 15 to 20 per cent. ferro, whereas in others the benefits are largely due to secondary causes such as slag fluidity and sulfur elimination. In view of our extremely limited supply of high-grade ore and due also to the fact that our reserves of low-grade ore are being consumed at a rapid rate, an effort should be made whenever possible to conserve manganese. The practice of working to a high residual manganese should be followed whenever practicable.

BASIC OPEN-HEARTH SLAGS FOR BLAST-FURNACE BURDENS

As the practice of using high-manganese basic pig iron has increased, there has also been an increase in the amount of manganese in basic open-hearth slag. It is customary in many steel plants to charge open-hearth slag in the blast furnaces to recover the manganese and iron, and obtain the fluxing value of the lime. Run-off slags are probably used more extensively than finishing slags. From the viewpoint of conserving a limited and vital resource, the practice of using open-hearth slags in the blast furnace is very commendable.

In 1925, 37,087,342 tons of basic open-hearth steel ingots and castings was produced in this country. Assuming the weight of slag is 10 per cent. of the weight of steel, it follows that about 3,700,000 tons of basic open-hearth slag accompanied this tonnage of open-hearth steel. To the writers' knowledge the information which would show the average percentage of manganese in our annual production of basic open-hearth slag has never been compiled. Cone²⁵ has compiled the analyses of basic open-hearth slags from 16 steel companies. The content of manganese ranges from 2.5 to 11.5 per cent. MnO, depending in a large measure on the kind of iron used. In calculating the total manganese oxide content

²³ C. L. Kinney, Jr.: *Op. cit.*, 163.

²⁴ C. L. Kinney, Jr.: *Op. cit.*, 136. (See discussion by V. J. Pazetti, pp. 169-70.)

²⁵ E. F. Cone: *American Steel Slags and Their Uses. Iron Age.* (April 12, 1917) 99, 896-900.

of basic open-hearth slags for 1913, he uses 5.5 per cent. as an average value for MnO. Due to the increased use of high-manganese iron it seems reasonable to assume that the manganese oxide content of open-hearth slags in 1925 would average 7.5 per cent.

$$3,700,000 \times 7.5 \text{ per cent. MnO} = 277,500 \text{ tons MnO.}$$

This figure shows that the potential value of basic open-hearth slags as a means of making high-manganese pig iron is large. It is recognized, however, that the phosphorus as well as manganese would move in a closed cycle when the open-hearth slag is charged into the blast furnace and would necessitate the use of manganiferous ores at times to get rid of the accumulative effect on phosphorus. The slag from acid open hearths, acid Bessemer converters, ferro and spiegel blast furnaces, and converters making manganese steels are not as important as basic open-hearth slags but should be used whenever practicable. Cone quotes one steel maker as follows:

It costs the open-hearth department about \$8 per car to waste open-hearth slag which is saved by using it in the blast furnace.

Operators are no doubt using open-hearth slags in most cases where it is commercially expedient to do so. However, exact information on this point is not available. A flow sheet giving the disposition of our annual consumption of manganese in the iron and steel industry would show whether any material saving of manganese could be made by modifications in practice. The information to make such a flow sheet would have to originate with the iron and steel operators.

FERRO FROM DOMESTIC ORES

Our supply of ferro-grade ores would meet annual requirements of the steel industry for about 2 years. Due to the fact that our supply of low-grade material is more adequate and also that 75 per cent. of this type of ore occurs in Minnesota and has the advantage of lake shipment to steel centers, it is an outstanding metallurgical problem to produce ferromanganese from Minnesota manganiferous iron ores. As long as manganese is indispensable in the art of steel production, a means should be provided to make our country independent of imported ores in times of duress. Moreover, a safeguard against arbitrary prices should also be provided. With these objects in view the Bureau of Mines in cooperation with the Minnesota School of Mines Experiment Station has undertaken the following experimental program.

METHODS UNDER INVESTIGATION FOR MAKING FERRO FROM DOMESTIC ORES

In order to make ferromanganese from manganiferous iron ores it is necessary to change proportions of manganese, iron and phosphorus

considerably. This can be accomplished by pyrometallurgical processes based upon the selective oxidation or selective reduction of these three elements. Flow sheets giving the details of three methods which are under investigation appear on pages 326 and 329. In general these methods involve usual steel furnaces with some modifications. The

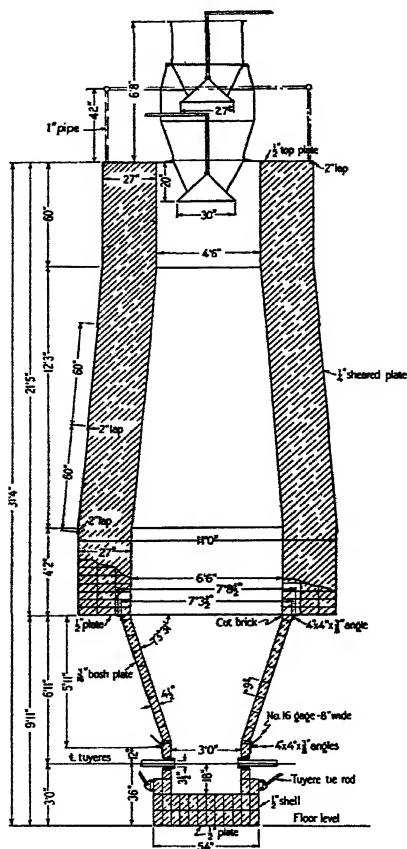


FIG. 5.—LINES OF EXPERIMENTAL BLAST FURNACE.

various operations will, we believe, be clear to those familiar with present iron and steel furnaces. Methods 1 and 2 involve the operation of a blast furnace upon a burden composed of 100 per cent. manganiferous ore. Although the brown manganiferous iron ores of the Cuyuna range are being smelted when mixed with other ores, the behavior of a burden composed of 100 per cent. of these ores was unknown. Two characteristics, high ratio of alumina to silica and large amounts of combined

water, suggested that these ores might differ in behavior from other lake ores. The cost of tests on an industrial furnace to determine the behavior of these ores was prohibitive. Inasmuch as an experimental blast furnace would be useful in acquiring fundamental data and in trying out new raw materials, several years were spent in developing the furnace shown in Figs. 5, 6, 7 and 8. A complete description of the furnace and

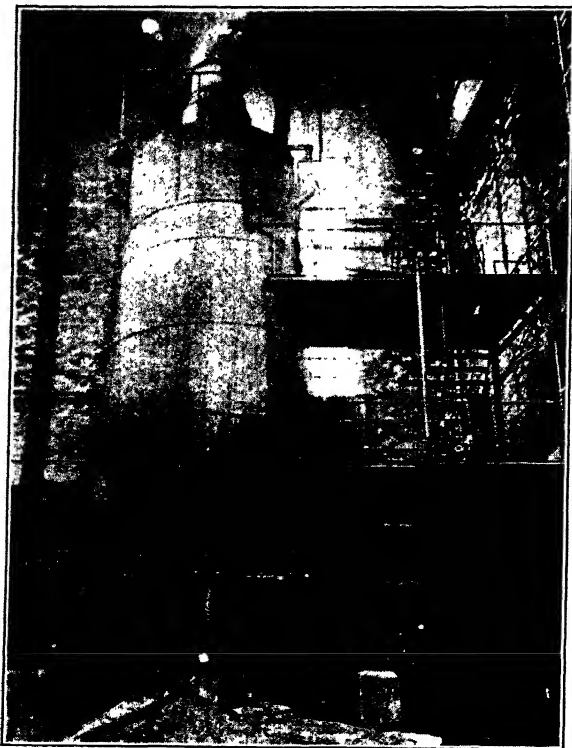


FIG. 6.—GENERAL VIEW OF EXPERIMENTAL FURNACE.

the results of a month's operation on brown Cuyuna ores are given in Bulletin 12, of the Minnesota School of Mines Experiment Station. On the basis of the experimental furnace data it seems reasonable to expect that a modern blast furnace with high blast heats could produce a ton of 15 per cent. manganese alloy on a fuel consumption well under 3000 lb. of coke per ton.

As the flow sheets for methods 1 and 2 indicate, the next step after melting the run-of-mine ore in the blast furnace is to treat the metal from the blast furnace in steel furnaces. In the oxidizing atmosphere

of the steel furnaces the manganese passes into the slag. The behavior of the phosphorus depends upon the kind of lining used.

Inasmuch as the experimental work has not progressed to the point where conclusions can be drawn, it is not possible at present to do more than describe the equipment which has been developed to study the problem. Just as it was necessary to develop a small blast furnace, it was necessary to develop a small open-hearth furnace. After operating

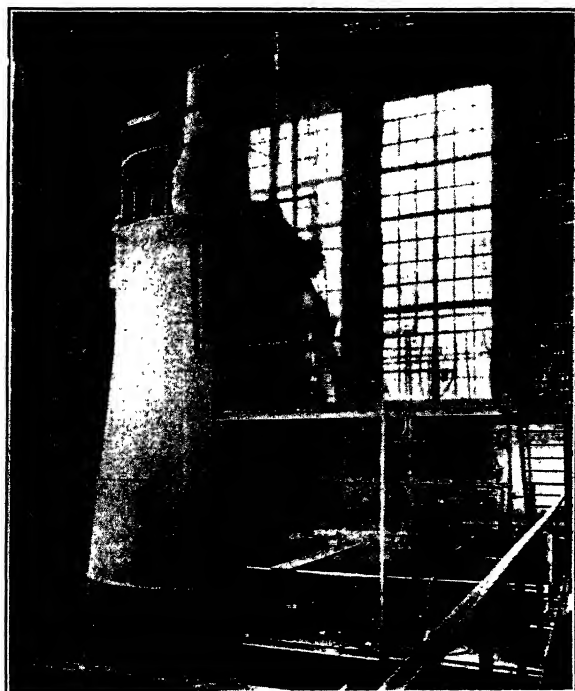


FIG. 7.—TOP SECTION OF EXPERIMENTAL FURNACE SHOWING SAMPLING PLATFORMS.

a smaller furnace the experimental open-hearth furnace, shown in Figs. 9, 10, 11 and 12, was erected. As brought out by Figs. 11 and 12, one important and perhaps the most important factor of this operation is the furnace lining. This problem is receiving most of our attention at the present time.

By referring to the flow sheet of method 2, it will be observed that the second step in this method is carried out in a converter. In practice the hot metal from the blast furnace or mixer would be charged directly into the furnace. However, in the experimental work it is not possible to run the converter and blast furnace simultaneously. As a result, it

will be necessary to melt the high-phosphorus spiegel (made in experimental blast furnace) in a cupola. Fig. 13 shows a cupola and a 600-lb. converter which will be used in investigating the possibilities of bessemerizing metal containing from 12 to 15 per cent. manganese, in order to recover the manganese in the slag. In the basic Bessemer process the silica, manganese, carbon and phosphorus are oxidized in the order named. The object of these experiments will be to determine the grade or type of slag which can be made by bessemerizing a speigel containing about 15 per cent. manganese and 0.5 per cent. phosphorus.



FIG. 8.—VIEW OF EXPERIMENTAL FURNACE AT CASTING.

Method 3, outlined on page 329, calls for a somewhat different procedure from methods 1 and 2. In this case the iron and manganese are separated by taking advantage of the fact that the iron ore can be reduced at a temperature of 950°C ., whereas manganese oxide requires a temperature approaching 1400°C . for rapid reduction. In place of the blast furnace the first step in this method consists of reducing the iron in the manganiferous ores by holding a reducing atmosphere at a temperature of about 900°C . Under these conditions the iron is metallized while the manganese remains practically unreduced. Metallization is

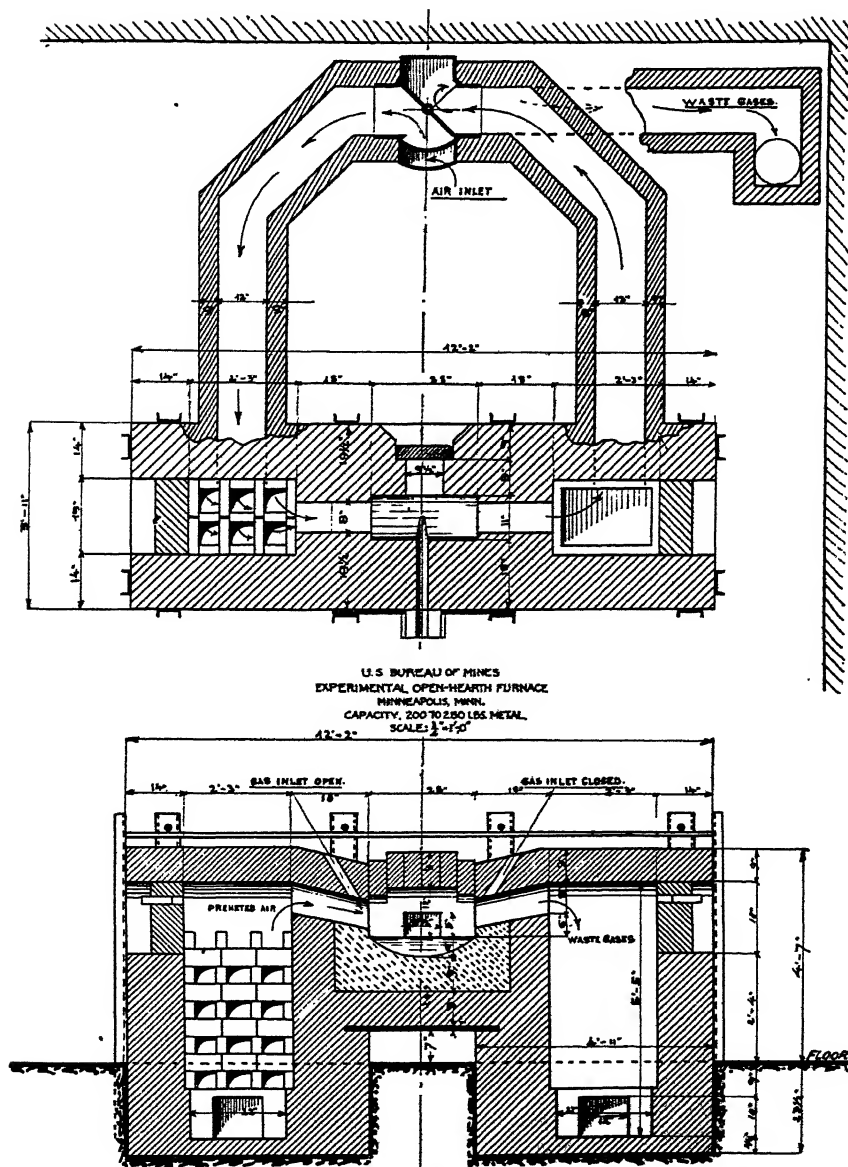


FIG. 9.—ELEVATION, PLAN, FLUES AND REVERSING VALVE OF EXPERIMENTAL OPEN-HEARTH FURNACE.

followed by cooling and magnetic concentration, if reduction is carried out in a furnace in which the ore is mixed with coal, coke, or other reducing agent. If reduction is carried out by circulating reducing gases through a shaft furnace, the hot material would be discharged from the shaft furnace into an electric furnace. Upon melting the unreduced manganese would pass into a slag, while the iron would be recovered as a high-phosphorus metal which could be further refined in the electric furnace or basic open hearth.

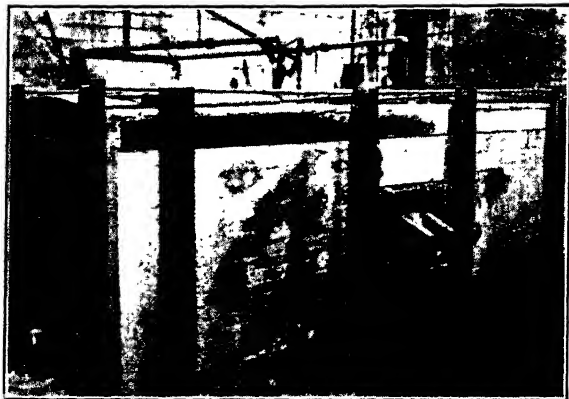
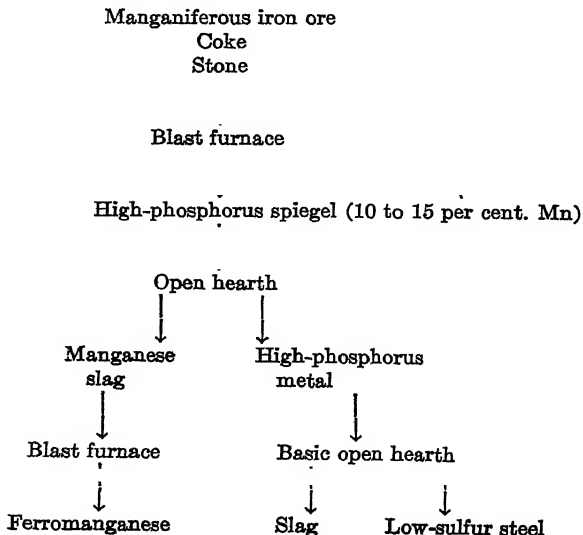


FIG. 10.—VIEW OF 200-LB. EXPERIMENTAL OPEN-HEARTH FURNACE.

FLOW SHEET.—METHOD NO. 1, FOR MAKING FERROMANGANESE FROM MANGANIFEROUS IRON ORE.



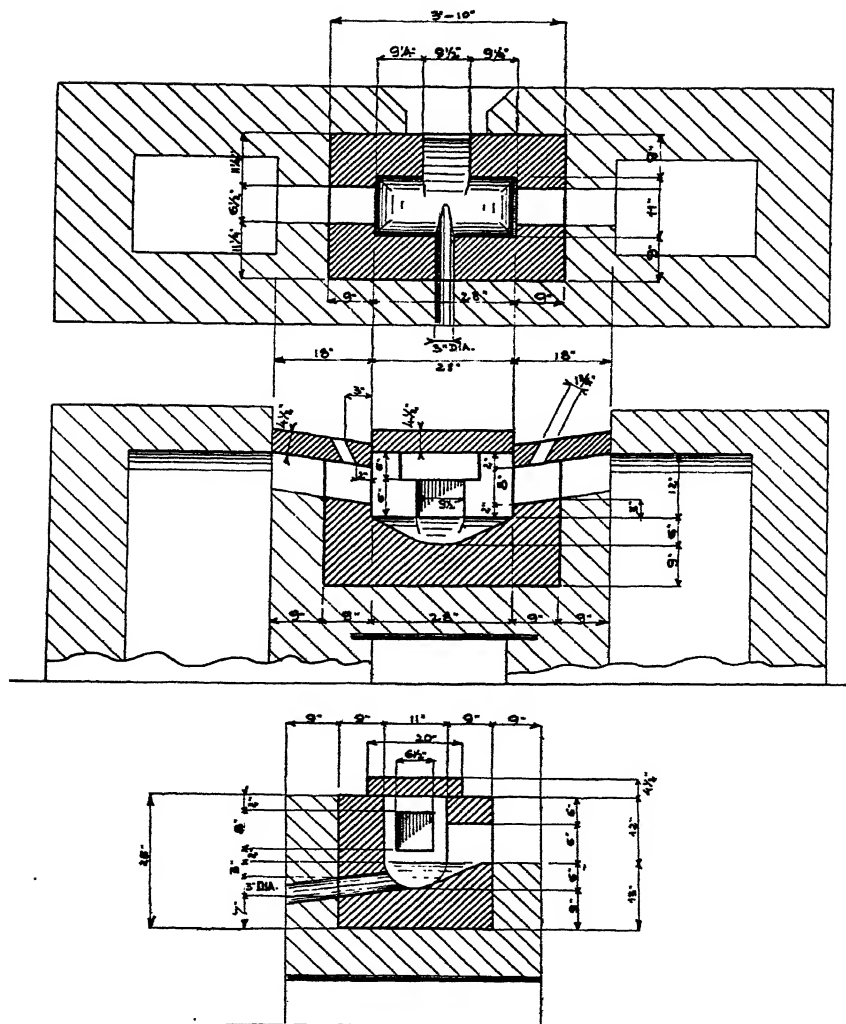


FIG. 11.—SECTION OF EXPERIMENTAL OPEN-HEARTH FURNACE LINED WITH VARIOUS REFRACTORIES. (SCALE: $\frac{3}{4}$ IN. = 1 FT.)

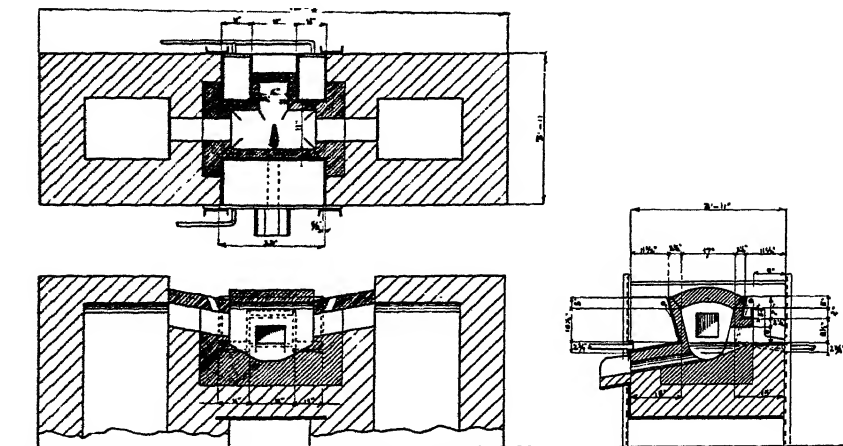


FIG. 12.—POSITION OF WATER-COOLING PLATES FOR WALLS OF EXPERIMENTAL OPEN-HEARTH FURNACE. (SCALE: $\frac{3}{4}$ IN. = 1 FT.)

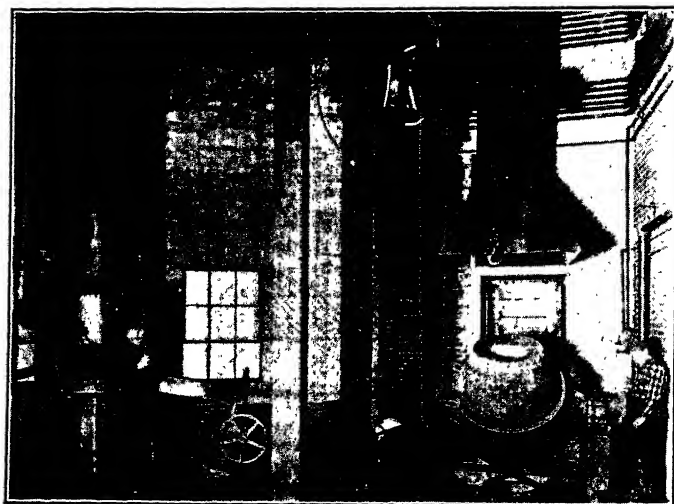
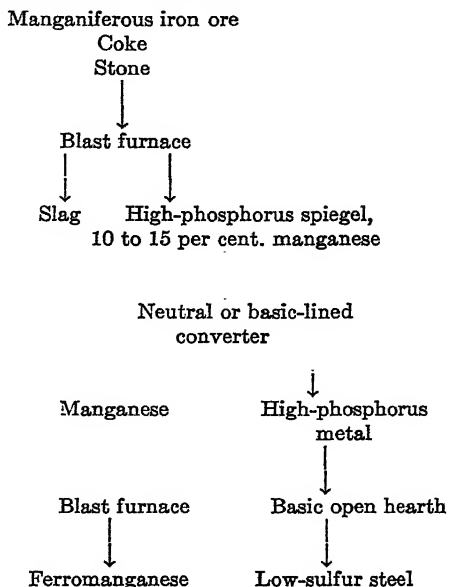
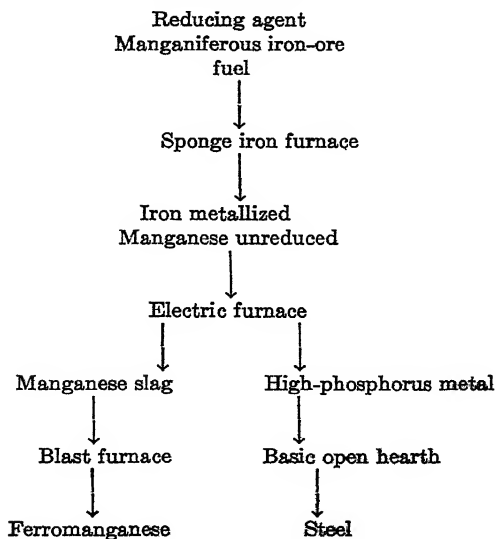


FIG. 13.—VIEW OF CUPOLA AND EXPERIMENTAL CONVERTER.

FLOW SHEET.—METHOD NO. 2, FOR MAKING FERRO FROM MANGANIFEROUS IRON ORE.



FLOW SHEET.—METHOD NO. 3, FOR MAKING FERRO FROM MANGANIFEROUS IRON ORE.



It has been suggested that the iron and manganese in the Cuyuna ores can be separated magnetically after the iron oxides have been metallized; laboratory tests²⁶ show, however, that a good separation is not possible because of the intimate association of the manganese and iron. In the Jones process the inventor proposes to separate the iron and manganese magnetically, grinding the sponge made from manganiferous ores and subsequently passing the fine material over a magnet. A microscopic examination²⁷ of samples of brown ore showed that

The maximum diameter of the well-defined veins of psilomelane rarely exceeds 0.16 mm. in diameter, or approximately 80 mesh, but the more intimately associated manganese oxide occurs in layers and forms of much smaller dimensions, many of them being smaller than the openings of a 200-mesh sieve (0.074) mm. Since the greater portion of the manganese in the samples examined appears to occur intimately associated in the manner noted, it would require fine crushing to secure liberation if the concentration of the iron and manganese into separate products were contemplated. Under the circumstances this would probably not be feasible.

Christianson and Hunter,²⁸ in an investigation of the Jones process, obtained results corroborating those of the writers which show that a good separation of iron and manganese is not possible by magnetic separation. However, by melting the sponge iron made at low temperatures, 93.5 per cent. of the iron, 2.1 per cent. of the manganese, and 88 per cent. of the phosphorus were recovered in the metal, while the slag contained 6.1 per cent. of the iron and 90.3 per cent. of the manganese.

Some preliminary experiments on this method have been made in a batch-type rotary kiln. These tests indicate that it will be difficult to magnetically separate the unused fuel and its ash in the rotary kiln product without the loss of considerable manganese. The work so far indicates that it will be very desirable to metallize the iron in these ores by circulating reducing gases through a column of the ore and discharging the product from the reducing chamber directly into an electric furnace. This entire procedure is essentially the same as direct processes for making steel, except that in this case a rich manganese slag would be recovered as a by-product. The 43 per cent. iron in the ore would be recovered in the form of steel, while the manganese would be retained in a slag from which ferromanganese could be produced.

It has not been possible to investigate these various methods and processes simultaneously. Progress has been slow, the method of attack being to get information on a laboratory scale whenever possible and proceed from laboratory tests to small-scale tests. The development and construction of various types of small furnaces has required consid-

²⁶ T. L. Joseph, P. H. Royster and S. P. Kinney: Utilization of Manganiferous Iron Ores. *Tech. Paper* 393, Bur. Mines (1926). See Table 5.

²⁷ Examination by R. E. Head, Bur. Mines, Salt Lake City, Utah.

²⁸ C. M. Weld and others: The Jones Process for Concentrating Manganese Ores, Results of Laboratory Investigations. *Bull.* 173, Bur. Mines. (1920) 71-91.

erable time. Although the work is far from completed and has only progressed to the point where it seems desirable to investigate certain phases of the problem on a larger scale, it seemed desirable due to the general interest in the domestic manganese situation to prepare a progress report giving the general aspects of the problem and the results obtained to date.

FURTHER DEVELOPMENT OF DOMESTIC RESERVES

The mineral reserves of the United States have been classified²⁹ as follows:

1. Minerals available in large quantities for export.
2. Minerals of adequate supply but without great excess or deficiency.
3. Minerals which exist in inadequate amounts.
4. Minerals almost entirely lacking.

Our present information regarding domestic reserves indicates clearly that manganese falls in Class 3. Development of domestic reserves of

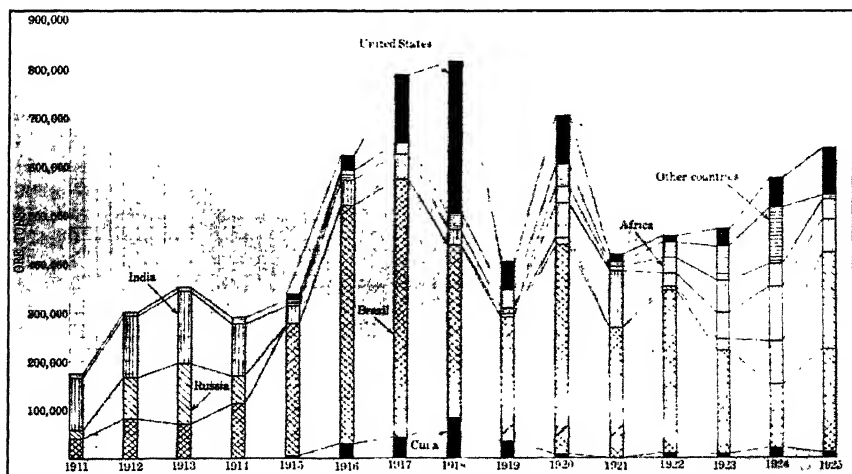


FIG. 14.—DOMESTIC PRODUCTION AND IMPORTS OF MANGANESE ORE.

minerals which fall in this classification will depend to a great extent upon the price demanded for imported material. Fig. 14 and Table 7 show that our four great sources of manganese are Russia, India, Brazil and the Gold Coast. From 1911 to 1913, our supply came from Brazil, Russia and India. The years 1914 to 1918 reflect the conditions resulting from the World War. Russian imports showed a decrease in 1914 and were stopped entirely from 1915 to 1918. Small tonnages were imported in 1919 and 1920. From 1922 to 1925 there was a gradual increase from

²⁹ International Control of Minerals. A. I. M. E. and M. & M. S. A. (1925) 11.

3400 to 194,006 tons. The years 1915 to 1918 show the gradual adjustment to the war situation, which caused increased consumption of manganese and almost complete reliance upon our own reserves and those from the nearest source of foreign supply, i. e., Brazil.

It is important to note that under the stimulation of high prices and modifications as to grade of material made necessary by war conditions, in general, we were only able to supply about one-third of our requirements from domestic reserves by 1918. The years 1919 to 1925 show the readjustment from war-time conditions and the tendency toward equalizing our imports from the various producing countries. At the present time the four major producing countries are competing for the world market. In regard to the present situation, Furness says:

TABLE 7.—*Imports and Domestic Production of Manganese Ore*¹

Year	Cuba	Brazil	Russia	India	Africa	Other Countries	United States ²
1911		41,600	19,103	106,580		9,568	2,457
1912		81,580	83,334	128,645		7,102	1,664
1913		70,200	124,337	147,587		2,966	4,048
1914		113,924	52,681	103,583		13,106	2,635
1915	5,141	268,786		36,450		10,401	9,558
1916	30,563	471,837		51,960		21,961	31,474
1917	44,511	512,517		48,975		23,969	129,351
1918	82,974	345,877		29,275		33,177	305,869
1919	35,320	246,592	6,916	9,200	1,338	33,948	54,957
1920	8,247	421,523	18,762	71,238	31,972	48,022	94,420
1921	34	262,468		113,730	8,453	7,621	13,531
1922 ³	12,568	327,000	3,400	24,846	28,699	28,495	13,404
1923 ³	9,062	203,000	24,200	51,136	61,400	70,202	31,500
1924 ³	23,065	126,000	85,000	109,132	49,000	112,803	56,515
1925 ⁴	12,745	207,238	194,006	68,326	33,582	3,242	97,500

¹ Figures from U. S. Department of Commerce.

² Tonnages include metallurgical and chemical ore containing 35 per cent. manganese or more.

³ Except for Cuba and United States, figures on manganese content are converted into tons of ore, using average manganese content for 1925.

⁴ Figures compiled by the U. S. Department of Commerce.

Reorganization of the manganese producers of the Caucasus and development of the Gold Coast deposits will unquestionably occasion a struggle to control the world's market. For many years India has supplied about 30 per cent. of the world's consumption and has maintained this ratio. Although the Caucasus has mined more cheaply than any other source, this slight advantage in cost has been offset by inferior physical qualities, and India and the Caucasus have competed most keenly. With the entrance of the Gold Coast into the world's markets, evidence points to the curtailment of the Brazilian and Indian output.

Costs in these three countries are so nearly identical that it is doubtful whether any one of them or a combination of any two could eliminate the third from supplying

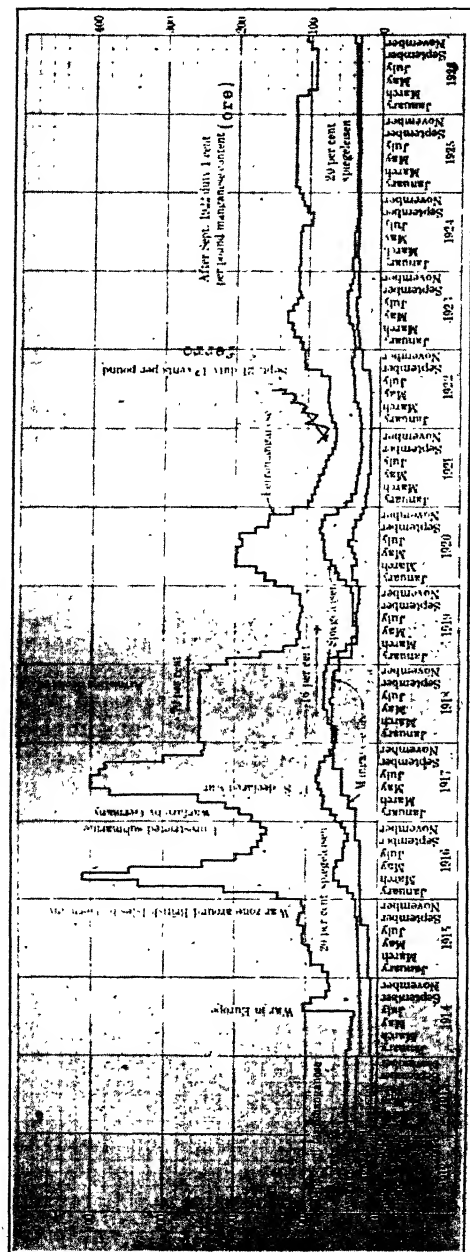


Fig. 15.—MONTHLY QUOTATIONS ON FERROMANGANESE, SPIEGELEISEN AND MANGANESE ORE, 1912 TO 1926.

part of the world's demand. Temporarily, at least, the Caucasus will be forced to reduce the price to insure sales of enough ore to satisfy the minimum royalty required in the agreement under which the concessionaires operate. In consequence, the selling price must fall to a point approaching that which existed before the World War.

As long as present conditions prevail and four countries are competing for the market, it seems unlikely that prices will permit utilization on a large scale of low-grade domestic ores in the production of ferromanganese. However, fundamental information should be obtained so that domestic ores will be available for the production of ferro in times of national emergency and at any future time when arbitrary prices may be demanded for imported ores.

PRICES OF FERROMANGANESE, SPIEGELEISEN AND MANGANESE ORE

Prior to the World War the price of manganese ore was about 20 c. per unit, or \$10 per ton for 50 per cent. ore. The price on the basis of 50 per cent. ore, c. i. f. North Atlantic ports, was \$12.50 per ton for the first 8 months of 1915. In July, 1918, the same quotation was \$68.50. Fig. 15³⁰ gives a graphical representation of the monthly quotations on ferromanganese, spiegeleisen and manganese ore from 1912 to 1926. If the tariff of \$11.20 per ton is subtracted from \$30.70, the price of the ore duty-free becomes \$19.50. Although prices of manganese ore have not declined to prewar levels, quotations have approached those existing before the War. During 1921, and the first 9 months of 1922, the average quotation on 50 per cent. ore at Atlantic seaboard was \$13.50 per ton. These low prices were partly due to small consumption of manganese ore as a result of small steel production and also to the fact that large stocks were carried over from the War. The main points of interest are that four countries are competing for the market and that the prices are fairly easy.

SILICOMANGANESE

The occurrence of large domestic deposits of siliceous manganese ore from which several grades of silicomanganese can be made has in the past suggested consideration of the possibilities of using manganese-silicon alloys in place of ferromanganese and ferrosilicon. The siliceous ores of Montana, Washington and California, and the siliceous manganiferous iron ores of Minnesota afford large quantities of ore from which a series of alloys could be made.

At the present time the true merits or disadvantages of silicomanganese are not definitely known because of its rather limited use. In 1919, E. F. Cone sent out a questionnaire to 100; representing foundries,

³⁰ Prices taken from *Iron Trade Review*. (Jan. 6, 1927) 90-91.

open-hearth, converter and electric furnaces, and steel foundries. The replies showed that silicomanganese was not in general use. Two companies expressed a preference for an alloy containing about 20 per cent. silicon and 50 to 55 per cent. manganese, but in general there was no agreement as to which composition was preferable. Several found the single alloy either uneconomical or unfit as a substitute for the usual deoxidizers. Others found it desirable but obtained no economy at the prices paid for silicomanganese.

Steel producers will be decidedly reluctant to use silicomanganese until there is an advantage either in price or improved practice. It is desirable in making low-carbon steels and particularly so when the manganese is high. Other advantages claimed for silicomanganese are fewer inclusions and greater efficiency of the single alloy. On the other hand, various objections have been made against the use of silicomanganese. The most important of these follow: it cannot be used for high-carbon, high-manganese steels; there is no saving in cost; the use of another alloy might confuse workmen; a single alloy decreases flexibility in operation; research and testing period necessary when making any change in practice; not desirable unless there is a tangible inducement to make the change.

The situation as a whole shows the need of a fundamental study to determine first whether the advantages claimed for silicomanganese are real or imaginary, and second what composition is most desirable. The economy which will result from the use of a single alloy will depend a great deal upon its actual advantages and the price of ferromanganese. While it is admitted that silicomanganese can not be substituted for ferromanganese and ferrosilicon in all cases, the problem should be investigated to determine to what extent we can rely upon silicomanganese and on domestic deposits of siliceous ore to combat arbitrary prices and conditions arising in times of a national emergency. If silicomanganese can be sold at prices about equal to those of ferromanganese, there is a fair chance that a market can be created in normal times.

SUMMARY AND CONCLUSIONS

1. Manganese is indispensable in the present art of steel making.
2. During the period 1911 to 1926, an average of 13.9 lb. of manganese in the form of ferro and spiegel was used in making a ton of steel.
3. Although the quantity of manganese derived from manganimiferous iron ores, used principally to produce basic pig iron, has increased from 14,629 tons of metallic manganese in 1911 to 147,500 tons in 1926, there has been no consistent decrease in the pounds of manganese required to make a ton of steel. The effect of higher manganese in basic iron has probably been masked by prices of ferro alloys, manganese specifications, and the increased use of steel scrap.

4. Domestic reserves of ferro grade ores (35 per cent. Mn or better) are totally inadequate. They would meet annual requirements for about 2 years.

5. It is estimated that our average annual manganese requirements in the steel industry for the next 20 years will be about 375,000 tons, and that 10 per cent. of this, or 37,500 tons, will be supplied from spiegel. If present tendencies continue, about 175,000 tons of metallic manganese will be derived from manganiferous ores. Converting these tonnages of metallic manganese into tons of domestic ore, the figures are as follows: for ferro, 1,000,000; spiegel, 400,000; pig iron, 2,000,000 tons.

6. It has been estimated that the total domestic tonnage of 5 to 35 per cent. manganese ores is from 35,000,000 to 48,000,000 tons. About 75 per cent. of the reserves are in Minnesota.

7. Minnesota deposits have the advantage of lake shipment to steel centers and relatively cheap mining costs.

8. Two general types of manganiferous ore occur in Minnesota. These have been designated as low-phosphorus high-silica ores or black ores and high-phosphorus or brown ores. Representative analyses of the two types of ore are submitted and their distribution shown according to composition. Considering what may be called proven ore, it is estimated there are about 3,500,000 tons of black ore (less than 20 per cent. SiO_2) and 25,000,000 tons of brown ore.

9. There are large deposits of black siliceous ores on the Cuyuna range which have not been regarded as ore due to the fact that they contain more than 20 per cent. silica. Men familiar with the range believe the ultimate tonnage of the Cuyuna range will greatly exceed the estimated figures.

10. Minnesota can supply the needs for high-manganese pig iron for 20 to 30 years, assuming there will be adjustments to compensate for some decline in quality.

11. The Cuyuna range manganiferous iron ores are not readily amenable to usual methods of beneficiation due to the fact that the iron, manganese and gangue are very finely disseminated.

12. Manganiferous iron ores when added to blast-furnace burdens to produce high-manganese basic iron aid desulfurization, particularly if large loads of sulfur are carried. The operation of the furnace is improved in several ways due largely to more fluid slags.

13. Manganese is desirable in basic iron because of sulfur elimination, better quality of steel, saving of ferromanganese, more fluid slags in the open hearth, less lime and fluorspar, faster working heats and increased tonnage. The advantages more than offset the disadvantages.

14. The amount of residual manganese varies from plant to plant. Some plants effect a saving of ferro when using high-manganese iron; in others the benefits are due largely to secondary causes such as slag

fluidity and sulfur elimination. The practice of working to a high-residual manganese should be encouraged whenever practicable.

15. A rough estimate indicates 3,700,000 tons of basic open-hearth slag, containing 277,500 tons of MnO , were produced in 1925. Such slags have a large potential value as a means of making high-manganese pig iron. Exact information is not available showing the extent to which open-hearth slags are used for making high-manganese pig iron.

16. A flow sheet giving the final disposition of our annual consumption of manganese by the iron and steel industry should reveal opportunities to save manganese by modifications in practice.

17. Inasmuch as manganese is indispensable in making good steel it is an outstanding metallurgical problem to produce ferromanganese from Minnesota manganiferous iron ores. A means should be provided which would make our country independent of imported ores in times of duress. Moreover, a safeguard against arbitrary prices should also be provided.

18. Several methods designed to produce ferromanganese from Minnesota manganiferous iron ores are under investigation by the Bureau of Mines in cooperation with the Minnesota School of Mines Experiment Station. Two of these methods are carried out with standard blast furnaces, converters and open-hearth furnaces. In order to test the different steps in the process on an experimental scale, an experimental blast furnace, open hearth and converter have been developed. A third pyrometallurgical method is similar to the so-called direct processes and is based upon the fact that manganese oxides require higher temperatures for reduction than iron oxides.

19. The manganese and iron in Cuyuna ores are too closely associated to be separated magnetically after the iron oxides have been metallized or converted to magnetite.

20. Further development of domestic reserves will depend upon prices demanded for imported ores. At the present time four countries, Russia, Brazil, India and Africa, are competing for the market and prices are fairly easy.

21. Silicomanganese can be made from domestic ores which occur in Minnesota, Montana, Washington and California. A fundamental study should be made to determine the relative merits of silicomanganese and the two alloys, ferromanganese and ferrosilicon. The most desirable composition should also be determined. It is believed that steel producers will use silicomanganese provided there is proper price inducement or an opportunity to make a better product.

22. Considerable information contained in this paper has been taken from a previous report, University of Minnesota School of Mines' *Bulletin* No. 12.

ACKNOWLEDGMENT

Credit is due Alexander Simonet for designing the experimental open hearth and converter.

DISCUSSION

C. K. LEITH.—Do you care to make any statement about the possibility of recovery of manganese from the siliceous ores, Mr. Joseph, or would Mr. Bradley want to make any statement on that?

T. L. JOSEPH.—Mr. Bradley can answer that better than I can. He is doing a very commendable piece of work. His efforts have been confined to the black ores, which are siliceous but contain more manganese than the brown ores. Silica is particularly objectionable in our metallurgical processes because it increases the slag, but silica which is not combined with manganese does not affect Mr. Bradley's process.

D. McLAIN, Milwaukee, Wis.—Do your calculations of the tonnage of manganese used in the steel include what is used in the iron foundries?

T. L. JOSEPH.—The amount used in steel only.

D. McLAIN.—During the past 25 years the average manganese content in gray iron practice has risen from 0.40 to 0.65 per cent., which is quite an increase. In making high-test cast iron and so-called semi-steel, as much as 3 per cent. of 80 per cent. ferromanganese is charged in the cupola when using 30 to 40 per cent. steel scrap for thin and medium castings. I wonder if you could not learn from the American Foundrymen's Association the probable tonnage of ferromanganese used in the iron foundry industry?

T. L. JOSEPH.—I think it would be a good thing to do. Some of the things I have brought out in the paper tend to show the necessity of checking the trend in the industry. From 1911 to 1922, statistics show very definitely that our manganese requirements were falling off. We are undoubtedly deriving considerable manganese from the manganiferous iron ore but the saving effected by this manganese is not known.

Several other things have come in; one is the ratio between the scrap and pig iron in the open-hearth charge. In 1911, we were making more pig iron than steel but now that is reversed. While we are actually getting more manganese in the pig iron, we are using less pig iron in open-hearth furnaces.

L. B. MILLER, Cleveland, Ohio (written discussion).—Quotation is made from a report of the Sub-Committee of the Mining and Metallurgical Society of America to the effect that under normal conditions the United States has practically no commercial high-grade manganese ores as they exist naturally and that their inquiry need go no further in this direction. To this statement, it seems that no substantial objection can be offered and their findings, in this respect, are largely accepted as correct. It seems likely, therefore, that supplies of material that possibly can be transformed into high-grade ore are the main source from which domestic requirements of high-grade ore can be obtained.

Under encouragement of the present tariff on manganese ore, a process which has not yet been put into commercial operation will be likely to prove of major interest in this connection. This is the leaching process for the beneficiation of ferruginous manganese ores of the Bradley-Fitch Co. of Minneapolis, Minn. Mr. Wilson Bradley has given me permission to make a preliminary statement regarding this process. I think he would have preferred to defer such a statement until his company can have

a plant in commercial operation, but a few remarks in regard to it should prove interesting at this juncture. The notes given herein will doubtless be subject to more or less change, as Mr. Bradley advised that while he considers the result of their experiments a complete success, they are still at work trying out some economies that look very promising. No attempt is made here to give chemical reactions.

This leaching process beneficiates oxides and carbonates of manganese. The ore to be treated is reduced to 80-mesh size, and is roasted at 450° C., which reduces the ore to manganese monoxide, or MnO. The leaching medium is a 10 per cent. solution of sulfate of manganese, transforming the oxide of manganese first to sulfate of manganese. This is then treated with free ammonia, producing manganese hydroxide, which, when ignited or sintered, results in high-grade manganese ore. Most of the oxide of iron and silica and practically all of the alumina, lime, magnesia, phosphorus and carbonate of barium are discarded as waste. The temperature of 450° C., at which the ore is first roasted, has been found by experiment to be the point at which the greatest discard of oxide of iron can be made; 95 per cent. of all the manganese in the original ore can be saved. The process of leaching requires about 2 hr. when the ore is reduced to 80-mesh size.

The shipping ore from this process will analyze 68 to 70 per cent. manganese in natural condition after being sintered. Aside from metallic iron in the original ore, practically all of the remaining impurities are insoluble in sulfate of ammonia. For instance, phosphoric acid is insoluble and fully 80 to 90 per cent. is eliminated; it passes off with the other discards.

The results of a test of ore from Crow Wing County, Minnesota, of the black variety, show:

	ORIGINAL ORE, PER CENT.	CONCENTRATE, PER CENT.
Iron.....	30.00	5.68
Silica.....	28.00	1.00
Manganese.....	15.00	70.80
Phosphorus.....	0.10	0.020

(Manganese monoxide may run 77.46 per cent.).

If rhodonite or bementite are to be beneficiated, apparently it must be done by some other process than the sulfate of ammonia method, as this solvent is not effective on silicates of manganese.

While some chemical grades of ore may be made, the large majority of material manufactured by this process will be offered to makers of ferromanganese.

Referring to the black and brown ores carrying manganese from the Cuyuna district in Minnesota, it is evident that of the black ore mentioned it will take from 4.5 to 5 tons averaging 15 per cent. manganese to make one ton of concentrates by the Bradley leaching process. Of the brown ores, averaging 9.50 per cent. in manganese, from 7 to 7.5 tons would be required to make one ton of concentrates.

On page 310, the authors refer to large reserves, to which attention is called by Carl Zapffe, of another variety of ore with silica ranging between 35 and 45 per cent. The manganese in this averages 10 to 12 per cent. and there is apparently no reason why it could not be utilized by the Bradley leaching process, and should be a prolific source of additional material suited for beneficiation into high-grade ore.

Most of the brown ores contain such a comparatively low percentage of manganese that it may be considered inadvisable to include them as economically suitable for treatment by the leaching process. Too many tons would be required to make one ton of concentrates. They are suitable for use in adding manganese to pig iron and likely are much better adapted for that purpose than to go to the extreme of attempting to leach them for their manganese. It should be observed, however, that if parts of the higher manganese brown ores be utilized as leaching material, the higher

phosphorus is not detrimental; it can be removed in the leaching process and will not go into the concentrates. This same remark applies to high silica ores; the only requirement is that manganese in these lean ores shall be high enough in percentage, namely 10 to 15 per cent. or higher, so that the operation of leaching may be economically practicable.

In addition to the black ores the siliceous manganiferous ores and a possible part of the brown ores from Crow Wing County, Minnesota, there are other substantial supplies of material that can be utilized by the leaching process. One of these is a supply that has been indirectly referred to in *Bulletin* No. 734 of the U. S. Geological Survey, by Hugh D. Miser and W. R. Crane of the U. S. Bureau of Mines, on manganese ores of the Batesville District in the State of Arkansas. This bulletin was issued in 1922. The report of the Sub-Committee of the Mining and Metallurgical Society has used figures about the same as Mr. Miser's on available high-grade material in this district. In Mr. Miser's report, which typically covered the district, but which actually covered the places where openings had been made, he referred often to brown and chocolate colored clays. These clays are residual from the solution of limestone, locally called Fernvale, and in addition to high-grade lump ore, carry oxides of ferruginous manganese ore in finely divided condition in quantity somewhere between twenty-five and fifty times as much as in the shape of high-grade lump ore. No practicable method for the winning of the fine ore from such residual clay has been devised heretofore.

In his report on manganese,³¹ J. W. Furness makes statements to the following effect:

"The development of residual clay deposits in Virginia, Tennessee, Georgia, Alabama and Arkansas, has not progressed because there is now no known economical method for their beneficiation. All mills erected during the world war period to treat residual clays were failures. Concentration as now practiced in Arkansas has reverted to the most primitive methods."

It will be of interest to Mr. Furness, and to those interested in supplies of ferruginous manganese ore, to know that, owing to encouragement of the present tariff, a process has been originated and put into practice for the economical separation of these finely divided oxides from residual clay in the Batesville district. The analysis of the ore recovered by this process is about as follows: Manganese, 30 per cent.; iron, 18 per cent.; silica, 10 per cent.; alumina, 4 per cent.; lime, 6 per cent.; sulfur, 0.025 per cent.; carbonate of barium, 3 per cent.; phosphorus, 0.75 per cent. In this material the phosphorus is so high that the ore in its present state is not suitable for making spiegeleisen, and the iron is so high that it is unsuitable for use in making ferromanganese.

The quantity available cannot be computed at this time owing to lack of actual exploration, but it is great enough to add organically to other present known sources of supply.

This material, like the black ore from Crow Wing County, Minnesota, can be made high grade by the Bradley leaching process, and is close enough to part at least of the consuming districts to be transported at reasonable rates of freight.

It will take about two tons of this beneficiated Arkansas ore to make one ton of high-grade concentrates, and discards would be about one-half of those from the Crow Wing County ores.

On pages 334 and 337 of the paper under discussion, the remark is made that means should be considered that would make our country independent of imported ores both in times of duress and as a safeguard against arbitrary prices that might sometime be demanded for imported ores. The Cuyuna district in Minnesota and the Batesville

³¹ Mineral Resources of the United States in 1925. U. S. Bureau of Mines (Jan. 7, 1927) I, 177.

district in Arkansas undoubtedly contain the largest deposits of ferruginous manganese ore available in the United States at the present time for use by this leaching treatment.

The mode of occurrence of the ore in both Minnesota and Arkansas, and the number of tons required to make one ton of finished product make it impossible for the finished product to be a low-cost material, but it is at least an avenue of supply upon which consumers of high-grade manganese ore in the United States may calculate.

The leaching process and the gleaning of ores from residual clay have been fostered and fanned into life by the fact that the protective tariff on manganese ores has been high enough to encourage the work. At low selling prices of manganese products, these processes would not have been originated and cannot now operate on unduly low financial returns.

Other states, where substantial quantities of ferruginous manganese ores are known to exist, can, without doubt, hugely supplement the supplies now known to exist in Minnesota and Arkansas. Supplies, however, of beneficiated high-grade manganese ore cannot be made available on the spur of the moment. Preparations in advance, requiring considerable time, are necessary in order to make such material promptly available when required for actual use.

C. K. LEITH.—Is it your impression that the present tariff is high enough to make your process possible?

L. B. MILLER.—I have not very much to say about this tariff. Everybody knows that it looks like a rather high rate. Now that you bring up the question, I am wondering whether 10 or 15 c. a ton reduction in the cost of the manufacture of steel in this country is a sufficient excuse for taking the tariff off manganese ore and shutting down on the possibilities of organizing a real production of high-grade manganese ore in this country to make a national asset.

D. F. HEWETT, Washington, D. C.—There have been times during the past ten years when it has seemed to me that any man who has the courage to put down in print that he thinks there are so many tons of material of a certain grade in a specified place is some kind of a fool. Yet it has been quite necessary for me, as well as for others, to do so!

The attempt on the part of the Geological Survey to make an exact measure of domestic resources of manganese goes back to prewar days. It began in 1916 when it seemed that war was inevitable. A group of us had a number of conferences and debates and finally did an enormous amount of field work. As a result of all this, we had the courage to subscribe our names to sundry estimates of reserves. In looking back over those first estimates, and the way they were made, I have realized that neither the methods of obtaining them nor of stating them were above criticism. Can we be blamed if, in the light of our experience, we are rather critical of such statements in general?

As engineers, I think we will agree that price is inherent in the term ore and that estimates of reserves which ignore the price at which ore may be produced mean very little. Manganese, as well as iron and many other metals, is widespread in the earth's crust, but all that need interest us at the moment are the local concentrations that can be mined at a profit under assumed prices. One can liken the reserves of iron minerals to an immense pyramid of broad base and low slope, divided into horizontal zones according to iron content. At the top are the rather small bodies of magnetite and pure hematite containing 60 to 70 per cent. iron. For every ton of such material, however, there appear to be many more tons, say 10, of that which contains 50 to 60 per cent. iron. Further, of the materials containing 40 to 50 per cent. iron, there are again many more tons, say 100, and if one cares to consider the reserves of material

which range from 0 to 10 per cent. iron, or which average 5 per cent., the base of the pyramid is the whole crust of the earth! In the light of the simile, it is, of course, apparent that the reserves of iron and manganese-bearing materials in the crust is enormous, but all that need interest us here are the richer parts that can be produced at a profit under assumed prices.

In making the estimates of reserves for the Institute Committee, we of the Geological Survey tried to keep the price factor in mind. In considering the reserves of known deposits, we had before us the records of the War Minerals Relief Commission showing the costs of operation and losses over the war period at several hundred mines. I am not surprised that those estimates are being criticized; I fully expected criticism and am glad to get it if it is constructive. Generally, the more criticism we get, the more data are brought out, and that is what we are after.

Now, realizing the criticism that would meet these estimates of reserves, we of the Committee felt that we should go farther and undertake to estimate what the future held in the way of production at price. Reserves might always be unprovable, but estimates of production at price are soon proved or disproved. It seemed to us that if we really knew as much about domestic reserves as we thought we did, we should be able to make some accurate forecasts of production. I am sorry that our critics ignore this part of the Committee's report.

Since the war we have all learned that the dollar is not a fixed unit and that its purchasing power fluctuates widely; roughly, in accordance with the general commodity index. Acting on the assumption that manganese ore will be produced in proportion to the ratio of manganese-ore prices to general commodity prices, we coined the term "manganese index number," which is the price of manganese ore divided by the current commodity index. On page 23 of the report,³² you will find that we estimated production of manganese ore for three successive years at three "manganese index numbers;" \$20, \$35 and \$50. At the time these estimates were made, April, 1923, we had before us the estimates of C. W. Potts, working for the American Mining Congress, that the reserves of domestic high-grade ore were 10,000,000 tons containing 42.5 per cent. manganese, and that under the tariff of September, 1922, domestic mines could supply the major part of domestic needs.

Our estimates of production are repeated here: for the first year at \$20 index number, 33,850 tons; second year, 51,100 tons; third year, 47,800 tons or a total of 132,800 tons for the three years. Production for 1922 had been about 13,000 tons. Fortunately, since the tariff was enacted in 1922, the manganese index number has not fluctuated 10 per cent. from the \$20 figure, and our estimates can be compared with actual production. The production for 1923 was 31,500 tons; for 1924, 56,515 tons, and for 1925, 93,324 tons. Of this 93,324 tons, however, 47,507 was rhodochrosite produced by the Emma mine in Butte, as a by-product of zinc mining. The mine cannot be worked profitably as a source of manganese alone, but now and then, when general operations for zinc develop reserves of manganese carbonate, it can be mined and shipped. If, from the total of 186,339 tons shipped during the 3 years 1923, 1924, 1925, the 47,507 tons be deducted, the remainder, representing high-grade oxide ore, is 138,832 tons, compared with 132,800 tons forecast three years in advance. I leave it to you whether the Committee is justified in having confidence in its estimates of production at price and reserves at price.

My attitude toward the Committee's report can be briefly summarized. I fully realize the difficulties that men of high ability and best intentions meet in attempting to estimate the mineral reserves of a country, a region, or even of a deposit. If such men are to try to reach an accord in these matters, the price factor must be considered.

³² Report on Manganese for U. S. War Department. A. I. M. E. Committee on Industrial Preparedness (Jan. 29, 1924).

The government needs, and has a right to expect from competent persons, opinions in such matters. Those of us who are consulted, and give our opinions, welcome constructive criticism, especially if it brings out new data, but hope that such criticism is given in good faith and on a basis on which reasonable persons can reach substantial agreement.

CARL ZAPFFE, Brainerd, Minn. (written discussion).—Although the work described by these authors may be summarized as an investigation of pyrometallurgical methods for making a high-grade ferromanganese alloy from Cuyuna manganese iron ores, the ores probably still constitute the main point for consideration. A reader of the paper would most likely infer that in such experiments the standard type of ore had been used, but one finds no definite statement about it. The paper contains many tabulations of analyses of ores from various Cuyuna properties.

On page 311, the statement occurs "that one problem lies in developing a process for the treatment of the brown ores which contain from 8 to 12 per cent. manganese and 0.2 to 0.3 per cent. phosphorus;" and the next applicable remark appears on page 321: "Two characteristics, high ratio of alumina to silica and large amounts of combined water, suggested that these ores might differ in behavior from other lake ore." We are given no other information regarding the ores actually used in the tests; but the authors refer the reader to the furnace tests described in *Bulletin 12* of the Minnesota School of Mines Experiment Station. That bulletin bears the same title as the paper under discussion, and Mr. Joseph is the senior author of both. The bulletin presents in its Table VI (pages 46 and 47) the analyses of the ore that was actually used. It was divided into 38 lots, and the analyses of the lots vary by very small values only. The average of all ores or lots listed figures:

	DRY, PER CENT.	NATURAL, PER CENT.
Iron.....	40.77	33.80
Phosphorus.....	0.31	
Silica.....	9.91	
Alumina.....	7.26	
Manganese.....	9.80	8.13
Ignition loss.....	11.80	
Original moisture.....	17.09	
Combined metallic units.....	50.57	41.93

I contend that this analysis shows the ore used to be of a very rare type, and that no operator of any existing mine could produce regularly more than a few carloads of it at any time. Only rarely could an ore with as much as 7 per cent. of alumina be produced; and I doubt if the above combination of silica plus alumina ever existed in shipments of consequence made heretofore.

I have been quoted by the authors in both papers, and my tabulations of analyses of Cuyuna ores published in my Institute paper of 1925 are reproduced by them. While some additions could now be made to these lists, no essential thing already shown would be different, and the preponderance of ores shipped to date would be shown to contain less than 4 per cent. alumina. These analyses are now shown as Tables 5 and 6 in their Institute paper, or Tables IV and V in their School of Mines bulletin.

Again, experience has shown that when a Cuyuna ore contains about 4 to 5 per cent. alumina, a distinct increase in moisture content follows. Therefore it is not to be wondered at that the ore used in the tests containing 7.26 per cent. alumina should have contained as much moisture as 17 per cent. and over. Some areas have ore even higher in ignition loss than the 11.80 per cent. quoted for the ore tested, but those ores

are prevailing low in silica and have no more than a moderate amount of alumina. This loss is not an important factor in a furnace operation. Certain high-moisture ores have the most desirable alumina-silica ratio.

To all concerned—users of ore as well as producers, and especially those who experiment with these ores—it should be emphasized that the ore that was used in these tests is neither typical, representative nor abundant.

It is not my intention to depreciate the work done or the results obtained; in fact, the work deserves greater emphasis than they gave it. We have the authors' assurance that the high alumina content "does not seriously militate against the smelting of these ores;" wherefore we may feel that the usual run of Cuyuna ores should be a boon to furnace operators. The application of the alumina-silica ratios quoted by the authors, and the slag compositions obtained, might likewise be misjudged; but these are not deemed serious by the authors, and that, too, should be emphasized for its value in other respects.

Cuyuna manganiferous ores as now produced are carefully graded as to uniformity in composition when making shipments and any wide variations from the normal ore is unusual; consequently, it should be made clear that these tests were made on an exceptional kind of ore, which adds greatly to the value of the results obtained by the authors. Neither is it correct to surmise that the tests contemplate using a certain future reserve of similar kind of ore, because the big reserve of the district is of a very different kind, chemically and physically.

T. L. JOSEPH (author's reply to discussion).—The authors did not intend to infer that the ore used in the experimental blast-furnace tests was typical in all respects of the brown Cuyuna manganiferous iron ores, and are glad to make a more complete statement of the points raised by Mr. Zapffe. Tables 5, 6 and 7, of *Bulletin* 12, referred to by Mr. Zapffe, show that the ore used was high in alumina and moisture. However, in regard to manganese and phosphorus content, drilling records indicate that there are about 20,000,000 tons of proven ore on the Cuyuna range which averages 0.27 per cent. phosphorus, and 9.5 per cent. manganese (see page 308). The amounts of manganese and phosphorus in the ore used approximate these values. Considering the experimental program as a whole, or up to the point where material suitable for producing ferromanganese is produced, the manganese and phosphorus are of greater importance than the alumina and moisture content, because the former affect the grade of alloy that can be produced and the latter only the first, or blast-furnace, step of the treatment.

One of the objects of the blast-furnace test was to show that the brown Cuyuna range ores on the whole can be used to make up a blast-furnace burden. Under present practice the brown ores are mixed with straight iron ores, no practice being available to show the behavior of a burden composed entirely of brown ores. A test using an ore approximating the average shipping analyses would show the behavior of a single ore of average composition, but would leave some uncertainty regarding the ores in the high range of alumina in proportion to silica. The effect of alumina on blast-furnace practice in general has been widely discussed. It therefore seemed advisable to lay emphasis on the percentage of alumina in the slag and to work in a range somewhat higher than is normally encountered. Many of the Cuyuna ores, which would produce slags in the normal range of alumina content, 10 to 15 per cent., present no problem.

The total amounts of silica and alumina determine the slag volume but the slag composition, which is also important, is largely determined by the ratio of these two substances in the ore. On an average, the ore used contained about $1\frac{1}{2}$ times as much silica as alumina. However, in test period 5 (page 48, *Bulletin* 12), the silica in the ore was about double the alumina, and the slag produced averaged 12.27 per cent,

alumina. Although the sum of the silica and alumina was high, the relative amounts are within the range of composition of the brown ores. In order that the results would have a wider application, an ore high in alumina was purposely selected and graded into lots to produce slag varying over a wide range in alumina content. Individual flushes varied from 9.40 to 21 per cent. alumina. As a whole, the slags were in the high range of alumina content, and the general operation of the furnace indicated that the maximum alumina and moisture content likely to be encountered in these ores will not prohibit smelting them without mixing with other ores. Although it is difficult to isolate one variable in the blast-furnace process, a résumé of the effect of varying percentages of alumina may be found on pages 61 to 64 of *Bulletin 12*, which contains more detailed information than could be included in the paper under discussion.

Mr. Miller has stated that the brown ores of the Cuyuna range are not in general suitable for treatment by the leaching process being developed by Mr. Bradley because they do not as a rule contain sufficient amounts of manganese (15 per cent.) to permit an economical operation. Due to the fact that there are large proven reserves of brown ore (20,000,000 tons), it is important to develop a method whereby their manganese content can be converted into the essential commodity ferromanganese. It is true that these ores can and are being used to make high-manganese pig iron. However, most of the manganese of this important reserve is by this use finding a way to irretrievable slag dumps.

The methods under investigation by the Bureau of Mines are designed to treat the brown ores, which contain substantial amounts of iron as well as manganese, in furnaces now used in the iron and steel industry. Under proper price stimulus for ferromanganese or acute shortage in times of duress, idle blast furnaces and open-hearth furnaces could be used to supply on a large scale this essential commodity.

Pyrometallurgical methods are not suitable for treating siliceous materials due to the large slag volumes and losses which would result from such treatment. On the other hand hydrometallurgical processes are not likely to prove economical unless the ore contains about 15 per cent. of manganese. Obviously there is a distinct field for both types of processes.

Reserves of Lake Superior Manganiferous Iron Ores

BY CARL ZAPFFE,* BRAINERD, MINN.

(Cleveland Meeting, April, 1927)

THE manganese ore reserves of Lake Superior, because of their location and nature, have recently achieved a marked degree of importance as compared with the world's manganese reserves. To appreciate fully the significance and the adequacy of this reserve, one should understand how such ores are used, where obtainable, the quantity used, possible avenues for increasing use or application, and the chemical and physical characteristics of the available ores. Only with those data at hand, is it possible to arrive at a convincing answer to the question whether the reserves of manganiferous iron ores are adequate to satisfy the annual demand of the presently known iron ores for the full life of the latter. That they should last is vital; and to ascertain whether they will last was the purpose of the present investigation, which marks the first attempt to study these Lake Superior reserves.

The Lake Superior region possesses and supplies the full assortment of iron ores needed for the conversion of ore into pig iron and steel ingots and castings. These ores are named bessemer, low-phosphorus and high-phosphorus non-bessemer, manganiferous and siliceous; and to include the term aluminiferous would be in keeping with the most recent nomenclature adopted by some dealers.

About 35 or 40 years ago, perhaps less, most of the Lake Superior ore which is being used today was the material then considered either useless or too poor for immediate use; perhaps it was considered no more than a future reserve. An examination of the analyses of ores and the guarantees in the sales of those older times convince one that the opinions of those days must have been governed by only small differences in analyses. Some grades of ores used today in substantial amounts were used most sparingly only one or two decades ago. The relative quantities of the different ores used are ever changing; and, bit by bit, the metallurgy and the opinions are ever changing and the prejudices against certain ores and practices decrease. This is unquestionably more pronounced for the manganiferous ores than for any other kind. If, when using the present data for manganiferous ores, we will only be content to make allowances for the same experiences other kinds of ores

* Manager, iron-ore properties, Northern Pacific Railway Co.

have had in those relatively short periods, it will be quite easy to apply rather simple mathematics to obtain a favorable and convincing answer for the main question.

The subject divides into two parts; one describes the uses, analyses, and occurrences of the ores, the other discusses tonnage estimates. The problem involves (1) judging the prospective future annual consumption, (2) offering a tonnage estimate of the reserves, and (3) then deciding whether this estimated reserve of manganiferous ore will last as long as the known estimated reserve of iron ore.

USES, ANALYSES AND OCCURRENCE OF THE ORES

Uses of Manganiferous Iron Ores

Manganese is used chiefly in the manufacture of steel, to a slight extent in foundry practice, and for making special alloys. It aids in the process of manufacturing steel. It imparts important physical properties to the finished steel when used as an alloy, such as ferro-manganese, spiegeleisen and manganiferous pig iron. The latter is a special pig iron and contains approximately from 4 to 10 per cent. manganese.

Generally speaking, unless the ratio of manganese to iron in the ore is right, neither ferro nor spiegel can be made by present practices; and as these alloys require fairly high manganese contents, it follows that low-manganese ores are today suitable only for mixing or are definitely consigned to the making of manganiferous pig iron. Because of the acknowledged scarcity of high-grade manganese ores in the United States and the complete lack of them in the Lake Superior region, we need here concern ourselves only with the so-called manganiferous iron ores.

Trend of Present Research

The use of manganiferous pig iron in open-hearth practice has often been discussed in recent years. Apparently the advantages outweigh the disadvantages, and much of what was formerly often considered disadvantageous has been proved fallacious or has been overcome.¹ The Joint Committee of the two mining and metallurgical societies, in its report² of July, 1925, recommended undertaking such studies as will lead to the substitution of ferromanganese by spiegel and high-manganese pig iron. Research of the utilization of manganiferous iron ores has been in progress

¹ E. A. Wheaton: Use of High-manganese Iron in Basic Open-hearth Practice. *Jnl. Iron Steel Inst.* (1920) 10, 395.

A. W. Smith: Use of High-manganese Basic Pig-iron and Manganiferous Ores in Open-hearth Steel Practice. *Jnl. Iron Steel Inst.* (1926) 16, 454.

² International Control of Minerals. Published jointly by the Mining and Metallurgical Society of America and American Institute of Mining & Metallurgical Engineers (1925) 51.

for some time at the Mines Experiment Station³ in Minneapolis, partly by the State and partly by the United States Bureau of Mines. That work is intended to determine such things as: the possible applications in general blast-furnace practice; the effect of high-manganese pig iron on the quality of steel; the possibility of decreasing the use of ferro-manganese and increasing the use of high-manganese pig iron instead; and the use of 6 to 10 per cent. alloys as additions to certain steels. The whole trend of thought today, therefore, is directed toward increased application, thus indicating that a greater consumption of these ores may be expected rather than a continued use in the present proportions.

Annual Shipments of Manganiferous Ore

Table 1 lists the tonnages shipped from the Lake Superior region since 1902, of all grades of ore, all manganiferous ores, and the percentage the latter tonnage bears to the total production. The tonnages have been obtained from the published records of the Lake Superior Iron Ore Association, but have been tabulated as shown, for the present specific purpose.

Producing Districts

The tonnages given as manganiferous iron ores in Table 1 are for the ores that have been used by consumers of Lake ores because of their manganese content. For many years the United States Geological Survey, the Bureau of Mines, and other collectors and disseminators of statistics, have been listing as "ferruginous manganese" and "manganiferous iron ores" for the United States those ores that analyze between 5 and 35 per cent. manganese. So far as the Lake Superior ore districts are concerned, the author ventures to state that there is no 35 per cent. ore in quantity; he also desires to call attention to the fact that ore as low as 2 per cent. manganese is kept separate and used each year because of the manganese content. Enough years have passed to convince us what the trade can and does use. We, of the Lake Superior region, have every reason for taking 2 per cent. ore into account as a lower limit, and as our upper limit for ores as mined has not been far above 10 per cent., existing estimates based on a range of 5 to 35 per cent. are misleading. An average of 2 per cent., as a yearly average from any one district, is the lowest average tabulated in over a quarter-century; but it should be added that it has applied to ore from only one district, has involved only 20,027 tons, and it happened in the disastrous year of 1921. We are actually well above that minimum limit. Consequently, we find productions credited to the Gogebic and

³ T. L. Joseph, P. L. Royster and S. P. Kinney: Utilization of Manganiferous Iron Ores. *Tech. Paper* 393, Bur. of Mines (1926).

TABLE 1.—*Table of Tonnages and Percentages of Lake Superior Ores 1902 to 1926 Inclusive*

	All Grades of Ore Shipped, Tons	Manganiferous Ore Only, Tons	Manganiferous Ore, Percentage of Total Tonnage
1926	*60,000,000	*2,500,000	4.2
1925	54,784,355	2,247,472	4.1
1924	43,276,165	1,438,241	3.3
1923	60,024,176	2,441,586	4.1
1922	43,431,294	1,592,266	3.7
1921	22,523,989	230,726	1.0
1920	57,938,428	2,377,560	4.1
1919	46,666,878	1,732,300	3.7
1918	61,972,871	2,613,135	4.2
1917	62,521,465	1,621,305	2.6
1916	65,191,903	1,794,325	2.8
1915	45,883,488	881,020	1.8
1914	31,434,096	622,290	1.9
1913	47,906,332	882,144	1.8
1912	45,863,223	827,154	1.8
1911	31,501,183	450,127	1.4
1910	42,366,180	650,491	1.5
1909	41,959,668	716,310	1.7
1908	25,744,084	264,884	
1907	40,229,789	651,468	
1906	37,600,858	717,049	1.9
1905	33,675,404	264,573	0.7
1904	21,435,988	145,195	0.7
1903	23,729,950	335,810	1.4
1902	26,777,101	269,131	1.0
Grand total.	1,074,438,868	28,266,562	*2.63

* Approximate (vessel and rail) and subject to correction.

^b Average for the period 1902 to 1917 inclusive, is only 1.6 per cent.

Average for the period 1918 to 1926 inclusive, is 3.6 per cent.

Menominee districts in Michigan and Wisconsin and the Mesabi and Cuyuna districts in Minnesota.

Mesabi District.—The productions since 1902 are platted by districts in Fig. 1. The tonnage produced from the Mesabi district has, as a whole, been almost negligible for 3 or 5 years. Only five times in 21 years did the output exceed 300,000 tons. The Mesabi is now apparently in a period of low production for this class of ore. For the last 10 years the output was 0.8 per cent. of the total of all ores produced from that district.

Menominee District.—The tonnages of manganiferous iron ores produced from the Menominee district have been irregular from year to year and never large; 526,523 tons, in 1923, was the largest total, and only

seven times in 25 years did the output exceed 300,000 tons. For the past 10 years the output was 5.8 per cent. of the total of all ores produced from that district.

Gogebic District.—The tonnages produced from the Gogebic district have gradually been becoming larger, and in 1925 almost reached the million mark. For the last 10 years the output was 9.6 per cent. of the total of all ores produced from that district.

Cuyuna District.—The Cuyuna district is younger and has been shipping manganiferous ores since 1913. The first 3 years show only nominal

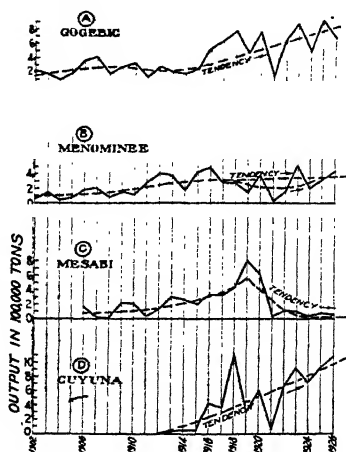


FIG. 1.—PRODUCTION CURVES FOR MANGANIFEROUS IRON ORES OF THE LAKE SUPERIOR REGION, 1902 TO 1926, INCLUSIVE.

quantities and together constitute 4.3 per cent. of the total of all ores shipped those 3 years. In 1918, 5 years later, the district shipped over one million tons, as it did again in 1926, the production for that year constituting over half of the total. The annual productions have varied between wide limits, but the tendency toward larger future outputs is a very marked feature. For the last 10 years the output was 34.8 per cent. of the total of all ores produced; during the last 4 years it was 50.5 per cent.

Analyses of Ores Produced

Table 2 gives the annual productions and the average manganese contents for the Mesabi, Gogebic, Menominee and Cuyuna districts for the years 1902 to 1925, inclusive. The figures of the Lake Superior Iron Ore Association are used.

Table 2 shows strikingly what each district has contributed annually and how the average analyses of the four differ from each other. The yearly analyses are given in Fig. 2.

TABLE 2.—*Tonnages and Manganese Content, by Years and Districts, for Lake Manganiferous Iron Ores Produced from 1902 to 1926*

	Gogebic		Menominee		Mesabi		Cuyuna	
	Tonnage	Mn, Per Cent.	Tonnage	Mn, Per Cent.	Tonnage	Mn, Per Cent.	Tonnage	Mn, Per Cent.
1926	682,897		433,765		43,620		1,088,218	
1925	933 318	4. 24	315,991	6. 70	72,250	4. 77	925,913	9. 02
1924	493,086	4. 00	198,963	8. 07	28,539	2. 52	717,653	8. 94
1923	878,976	5. 01	526,525	4. 51	104,282	3. 05	931,803	9. 04
1922	666,773	3. 85	155,146	5. 74	114,242	2. 98	656,105	8. 49
1921	141,259	2. 81	13,476	4. 00	20,027	2. 01	55,964	10. 64
1920	762,984	3. 76	388,393	4. 67	621,477	2. 29	604,706	10. 46
1919	477,964	3. 13	151,113	7. 76	807,708	2. 41	295,515	9. 88
1918	785,865	3. 24	280,502	5. 35	426,554	3. 27	1,120,214	10. 56
1917	654,006	4. 00	280,682	6. 80	321,395	3. 20	365,222	12. 21
1916	545,550	3. 66	495,712	4. 01	333,595	2. 49	427,370	6. 99
1915	256,588	3. 52	427,630	3. 90	196,802	2. 34	45,048	13. 50
1914	182,554	3. 10	170,274	3. 69	269,462	3. 96	51,292	15. 01
1913	208,310	3. 13	375,037	3. 61	298,797	2. 77	24,434	14. 72
1912	282,124	2. 79	411,125	3. 37	133,905	2. 43		
1911	138,446	2. 95	285,662	3. 49	26,019	2. 20		
1910	326,717	3. 93	118,659	3. 21	205,115	2. 35		
1909	288,464	4. 92	147,523	2. 93	214,034	2. 62		
1908	183,511	3. 87	80,257	5. 03				
1907	421,106	3. 88	210,762	4. 82	19,600	4. 94		
1906	363,254	3. 66	186,502	5. 75	167,293	2. 42		
1905	194,497	4. 64	70,076	4. 25				
1904	99,857	3. 83	45,338	4. 00				
1903	178,653	5. 20	157,157	4. 00				
1902	222,141	3. 54	46,990	4. 71				
Grand total production.....	10,368,900		5,973,260		4,424,716		7,309,458	
Average annual production.....	414,756		238,930		221,236		522,104	
Average last 5 years.	731,010		326,078		72,587		883,938	
Weighted average analyses.....		3. 87		4. 72		2. 86		11. 32
		Average since 1917		5. 83		Average since 1918		10. 32
		Average prior to 1917.....		3. 92				

* Not final for the season.

The Mesabi ore is lowest in manganese, and the content has been unusually uniform. The whole output averages 2.86 per cent. manganese, it has never exceeded 4.94 per cent., and has exceeded 4 per cent. but twice. The two bigger productions in 1919 and 1920, followed by 6 years of small productions, appear as short spurts. An examination of the records of all the Mesabi shipments for 1918 and successive years has disclosed that in each year since 1920 a very large production of ore was graded at the mines as manganiferous ore; but because it ranged between 1.93 and 1.99 per cent. in manganese, it was not included by the Ore Association, which uses 2 per cent. as a lower limit.

The Gogebic ore has also been markedly uniform and averages only a little higher than the Mesabi ore, being 3.78 per cent. It has never exceeded 5.20 per cent. and has exceeded 5 per cent. but twice. Since 1923, the Gogebic ore has been averaging just a trifle above the general average.

The Menominee ore, as a whole, averages a little higher in manganese than the Gogebic ore, being 4.72 per cent. for the same 25-yr. period. It should be noted that prior to 1917 the analyses were lower than since that date, averaging 3.92 per cent. and 5.83 per cent. respectively (weighted averages). The average has never exceeded 8.07 per cent., and it exceeded 5 per cent. but eight times. A noticeable variation of

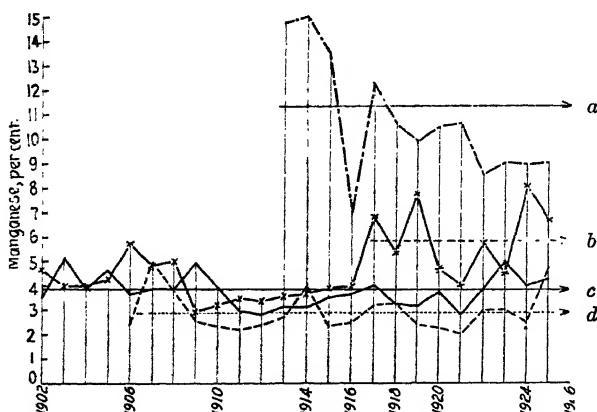


FIG. 2.—CURVES SHOWING AVERAGE MANGANESE CONTENT OF GOGEBIC, MENOMINEE, MESABI AND CUYUNA MANGANIFEROUS ORES, 1902 TO 1925, INCLUSIVE.

a, Cuyuna ore (average: 11.32 per cent.); *b*, Menominee ore (average: since 1917, 5.83 per cent.; prior to 1917, 3.92 per cent.; total, 4.72 per cent.); *c*, Gogebic ore, 3.87 per cent.; *d*, Mesabi ore, 2.86 per cent. The arrows indicate weighted averages for the whole period for each ore.

both tonnage and analysis, with a decided tendency for the higher analyses to be associated with the smaller tonnages produced, suggests that selective mining may have been practiced on special occasions; but some recent discoveries and greater developments in manganiferous ores are responsible in part for the better showing since 1917.

The Cuyuna district has a short record to offer, beginning with 1913. The ore produced that year was small in quantity, but averaged 14.72 per cent. in manganese and came from one mine, as was also the case in 1914. In 1918, 5 years later, 18 mines produced such ore, the production totalling 1,120,214 tons and averaging 10.56 per cent. in manganese. No such quantity had ever before been produced in any district, and the only higher analysis obtained, with 14 mines then producing, was in 1917, the average being 12.21 per cent. The 1926 production is tentatively

placed at 1,088,218 tons (exclusive of all-rail shipments) by the Ore Association; but private records show that the tonnage actually mined was 1,128,909 tons, and that it constituted as much as the combined total of all the other districts. The weighted average for the whole production is 11.32 per cent.; however, the whole tendency now is toward a production that is likely to yield a net result each year averaging around 9 per cent., as is already indicated by the analyses since 1918.

Present Methods of Beneficiation

As methods of beneficiation of iron ores we usually list crushing, screening, washing, jigging, drying and sintering. Crushing and screening benefit the physical structure of the iron ores. Drying and sintering do not remove ingredients objectionable to furnace operators, although sintering improves the structure if the ores are fine; drying might increase the fines. Washing and jigging are the only methods that are primarily designed to remove objectionable ingredients and make a good ore out of material that would otherwise be without immediate value. Because so little material is jigged, it is inevitable that when referring to beneficiation one's thoughts naturally center on washing.

To date, experiments with jigs have not shown sufficient improvement in the manganiferous ores that have been tested to make that method likely of widespread adoption for some time. Sintering is used at one mine, but it has not yet added materially to the aggregate production. Drying has been in vogue at one mine for several years, but not all ores produced there are dried. Crushing is used regularly at several mines because the ore is very lumpy when mined. Washing was tried twice and was abandoned. In short, no method of beneficiation has been applied regularly or continuously to improve the composition of these manganiferous iron ores. We may say that, as a whole, the ores produced are being used as they are found in their natural state.

In 1925, the author⁴ referred to large areas of manganiferous formation in the Cuyuna district containing 25 to 35 per cent. silica and more manganese than any Lake Superior ores yet produced, and stated, furthermore, that that material would not always be shunned and that the problem of making it usable would be attacked in due time. Since that statement was made, it has come to light that large experimental work is under way on that very material and much good progress has been made. The time is not yet ripe for any description of the work done or the results obtained. The process, which involves leaching the manganese with ammonia solutions, promises success. This will be an incentive for an attack of the problem along other lines as well as an expansion of the method thus far undertaken.

⁴ Carl Zapffe: Manganiferous Iron Ores of Cuyuna District, Minnesota. *Trans.* (1925) 71, 372.

Not included under the term beneficiation, but yielding a most important result, is a recent practice to provide suitable manganiferous ore by mixing different kinds of ores. If processes that affect only the physical structure are listed as modes of beneficiation, how much more appropriate it would be to include mixing widely different ores that affect the product analytically, just as washing does. Mixing is in a sense the reverse of washing. The latter process takes out undesirable material to make a commercial ore of non-usable material, and the former combines materials not usable alone but very desirable and very important as a mix. This procedure, practiced elsewhere in a smaller way, is referred to as grading and it is commonly deemed a function of mining. Conducted in a large way, it may constitute distinctly separate operations, and in importance is on a par with any process of beneficiation as usually described.

The best known and widest applied case of mixing ores is that of the Oliver Iron Mining Co., as used in the Mesabi district. It applies to iron ores. For many years this company has had 10 groups to represent its whole output from two dozen or more mines. In 1923 one group⁵ consisted of a mixture involving 18 properties. Probably not all those properties were always included. It is wholly possible that every bit of ore that goes into some of these group mixtures is useful even by itself; but it does not seem likely that in so extensive an operation as conducted by this company all separate quantities included in some groups would ever be disposed of except when mixed with other ores. This illustrates what can be done on a larger scale. It is not strictly what the writer has in mind.

A pertinent case in point is the mixing recently adopted by C. K. Quinn & Co. in handling the manganiferous ores from its group of Cuyuna mines. Four mines, known as Mahnomen, Mangan No. 2, Sultana and Hopkins, operating separately under different owners did not progress successfully and all but the original Mahnomen mine were abandoned by their operators. The Mahnomen mine continued very nicely in many respects, but it was postponing an obviously immediate problem, which would eventually block the whole operation. One by one these four properties were brought under one control and now constitute two separate open-pit mines. No attempts will be made here to discuss the details of the operator's problems, but he has pointed the way; and as it is essential that the point to be emphasized be set forth clearly, this statement is made: Three materials of doubtful value alone were taken from adjacent mines, the one material being high in manganese and silica and low in iron, another being low in manganese and silica, and another being relatively low in manganese, moderate in silica but high in iron; all were dumped in proper proportions into a crusher and again

⁵ Crowell and Murray: Iron Ores of Lake Superior. (5th ed. 1923.)

mixed upon being dumped into the ore dock at Superior, which is a feasible combination, possible of many variations, and results in a very desirable and salable ore. What has been done at the Mahanomen mine may be done elsewhere. The idea was simple. The obstacles were physical. If the idea is applied elsewhere and the physical obstacles surmounted, the procedure may be expanded without limit. In the Cuyuna district are many chances for application.

Summary

Considerable detail to make a complete story of fundamental facts, has been deemed advisable, although the facts may have been obvious to many for a long time and now present nothing new. However, in the use of these ores there has been a definite and notable trend, accompanied by a widespread interest. Their use in the iron and steel industry dates back farther than our tabulated records. The function of manganese has been known for ages, but recent years have developed a technique in furnace operations that has made the acquisition and the use of these ores mandatory, and it has raised them out of the class of off-grade iron ores. They now constitute an important single class. Although not mined primarily for the iron they contain, present practice in using the manganese makes a sufficient iron content in the ore an essential factor, wherefore they continue to be considered and sold in much the same manner as are the standard or well-known grades of iron ore of old.

In review, we find now these well established facts. (1) The manganiferous ores of the Lake Superior region have a definite place in iron and steel metallurgy. (2) The ores available have spurred experimenters into fields of research to seek results that are expected to lead to increased use of these ores, to make their uses of greater value to the steel industry, and to make them a greater protection to our country in time of war when foreign supplies of high-grade ores are unavailable. (3) With a rise in consumption from 26,000,000 tons of iron ore in 1902 to 60,000,000 tons in 1926, there occurred a rise in use of manganiferous iron ore from 269,000 tons to about 2,500,000 tons in the same period. But more significant is the increase in the percentage used, for whereas in 1902 the manganiferous ore constituted 1 per cent. of all ores produced, it now constitutes 4 per cent. (4) We find the ores available from 4 of the 6 established districts in the Lake Superior region. Two of the 4 districts have been producing uniformly since before 1902, and show improvements and gains in certain respects. One district, beginning to produce in 1906 and never abundantly, lately slackened in its production. Another district, the newest one of all, made its entrance upon the records with big annual productions, and after a very short time was found producing as much annually as the other three districts combined. (5) The district that has been relatively unimportant from the standpoint of production has

also been the producer of the ore with the lowest content of manganese. The newest district, and with the greatest output, is producing ore with an average content about twice that of the ores from the other districts. This, then, resolves itself into two major groups of manganiferous ores, the one averaging about 4 per cent. and the other 8 to 10 per cent. in manganese, and the greatest increase in demand is for the latter. (6) Standard beneficiation methods applied to improve materially the grades analytically are not yet in practice, but certain experimental work is being prosecuted vigorously. Mixing widely different ores has proved successful commercially.

This brings us face to face with a vital economic question. With the iron ore reserves quite definitely blocked out, and with manganiferous ores not well blocked out and facing an increasing demand, are the reserves of the latter sufficient to meet even the ratio of present requirements for the same length of time as the known iron ores will last? And how much greater a demand will the manganiferous ore reserves respond to as a result of the apparent increasing applications. We have good data for a discussion of those points, and for making a test calculation.

ORE RESERVES

A proper question is: Whose estimate should be used, or whose estimates correctly portray the situation? Measured ore reserves always change, due to cost of production, market values and usages. Many events in the last 20 years have curtailed explorations and modified methods of estimating. If one attempts now to draw conclusions from presently existing and published figures, he should have considerable familiarity with the state of existing explorations, the purposes for which various estimates have been made and the methods employed, and some knowledge as to the consensus of opinion concerning the value of existing exploration and other explorable areas. This is hardly within the powers of any single individual. It is here that we may expect a wide divergence of opinions. It seems necessary that a liberal view should be taken, if one would expect to make headway. Several parties may differ widely regarding how much weight or consideration should be given to individual properties or prospects; but where many properties and areas are involved, the aggregate result should give at least a fairly good portrayal of what confronts us. We will profit by first examining published estimates.

Iron Ore Reserves

M. C. Lake,⁶ in his comprehensive summary of Lake Superior iron ore reserves, made in 1925, tabulated figures as of Jan. 1, 1920, prepared by

⁶ M. C. Lake: The Future of the Lake Superior District. Lake Superior Mining Inst. (Aug., 1925) 24.

R. C. Allen, containing also estimates made by the Minnesota and Michigan Tax Commissions, C. K. Leith, W. O. Hotchkiss and W. G. Swart. The total of assured ore of all grades was, on Jan. 1, 1920, 1,560,505,305 tons; probable ore, 1,368,719,800 tons, and grand total, 2,947,225,108 tons. If we deduct therefrom the shipments for the years 1920 to 1926, inclusive, Lake's totals are reduced to:

	TONS
Assured Ore.....	1,213,072,457
Probable Ore.....	1,368,719,800
Grand Total as of Jan. 1, 1927.....	2,581,792,257

L. P. Barrett,⁷ Appraiser of Mines, State Tax Department of Michigan, gives a briefer summation based on State Tax Commission figures for Minnesota and Michigan:

	TONS
Active or Idle Equipped Mines.....	909,376,846
Reserve Tonnage.....	613,450,305
Total Reserves.....	1,522,827,151

This is exclusive of figures for Wisconsin. Inasmuch as Barrett used the Minnesota School of Mines Mining Directory of 1925 as his source of information for Minnesota, his estimate shows ore as of May 1, 1924. Deduction of shipments for 1924, 1925 and 1926, reduces Barrett's total reserve to about 1,350,000,000 tons, which is but little more than the corrected figure of 1,213,072,459, based on Lake's estimate as of 1920. In Michigan new ore has been added each year in nearly the amounts extracted.

Lake figured that in 1924 the assured tonnage he offered was good for 21 years; if no increase in output occurred, 27 years. The probable ore listed included some obvious speculative possibilities and it might be smaller in tonnage than the assured ore of today. The next 25 years may introduce many unlooked-for changes of various types which may even increase these figures. On the other hand, the assured ore is surrounded with less doubtful features and may be regarded as a definite and reliable quantity. It is generally agreed that the best possibilities for the addition of large tonnages of new iron ore are now manifest or even partly defined, and that the less favorable possibilities will be expensive to test and will attract attention slowly.

These tonnages quoted include, of course, all the manganiferous ores. By doing no more than rounding off the totals of the assured tonnage, to correct it for manganiferous ore included therein, the remaining assured iron ore as of today may be taken as 1,200,000,000 tons. That can hardly be open to serious objection.

⁷L. P. Barrett: Michigan Iron Mining Industry. Lake Superior Mining Inst. (Aug., 1925) 24.

Tonnage Requirements of Manganiferous Ores

The purpose of the above figures is to arrive at some idea of the tonnage of manganiferous ore needed to satisfy the tonnage of iron ore which we are going to consume in the next 20 years or so. It has been shown that at present 4 per cent. of the total output is manganiferous ore. Therefore, to satisfy 1,200,000,000 tons of iron ore during the next 20 or 25 years, we will need, with that ratio, 48,000,000 tons of manganiferous ore.

Just how much the increase will be over the 4 per cent. figure used will depend on factors not yet fully indicated. It is certain that the biggest part of the increase from 1 per cent. used in 1902, to 4 per cent. used in 1925 and 1926, resulted from the replacing of the bessemer process by the open-hearth process and the greater use of scrap iron; but we do not know now whether the latter will completely replace the former, or whether some other process will be developed which will similarly aug-

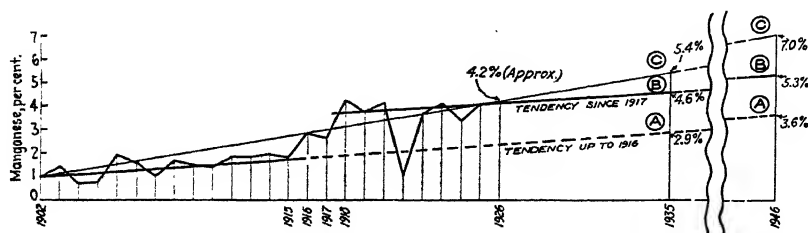


FIG. 3.—PROPORTION OF MANGANIFEROUS IRON ORES PRODUCED FROM 1902 TO 1926, INCLUSIVE, BY YEARS.

ment the consumption of manganiferous ore. The Bureau of Mines is experimenting to ascertain if spiegel and low-grade ferro can be made from Cuyuna ores. Should it be successful, there might perhaps be a substantial additional increase. Interested parties seem to feel that an increased demand is actually developing now from only ordinary developments in furnace practice. Making spiegel only, and not ferro, would not make much of an inroad on the reserves because little spiegel is used; but if low-grade ferro can also be made and found usable, and its use proves economically feasible and practical, then the demand will be much larger.

It is important that some data should be formulated pertaining to future consumption. We may surmise that there will be a total requirement greater than the 48,000,000 tons just mentioned. Without some such final figure determined, the whole study would lose much of its value. We want some idea regarding this figure; and whatever figure may be proposed, some justification should be advanced for it. True, any method used will be a speculative method, and the results will

be subject to some debate; we are not, however, without considerable reliable statistics that point the way, and, as well, give value to the conclusions derived therefrom.

Just as we have long used curves superimposed on past productions for iron ore to arrive at likely future productions, let us attempt the same with manganiferous ores. Fig. 3 shows percentages listed in Table 1; that is, the percentage that the annual output of manganiferous ore produced in any specific year bore to the total of all grades produced that year. That plat discloses some very important and very significant information. It is immediately evident that from 1902 to 1916 a moderate increase occurred for that period of years, as suggested by the tendency-line *A*. Since 1918, the tendency is represented by the line *B*. The offset between *A* and *B* is very abrupt. A pronounced increase in output happened between 1915 and 1918; but since then, the annual increase appears to be going along at virtually the same rate as before. That is evidenced by the fact that the two tendency-lines *A* and *B* are parallel. The ratio has not, therefore, been increasing during the recent years at a more rapid rate than it did in the earlier periods. If we joined the points for only the years 1902 and 1926, the tendency line would be like line *C*.

If we continue with the idea that the present uses or applications of manganiferous ore will be the dominant ones for many more years, then our conclusions as to future productions, when determined from curves developed from records of many years, should not be far out of the way. Linked with any output is the matter of manganese content; these ores are mined for the manganese they contain, and if the average content were decreased, the tonnage consumed would be increased to provide the additional units. Thereby our calculations might be upset; but, on second thought, that seems to be something not of the immediate future if present indications are a reliable guide. There are other factors, too; but with only little thought given to those features, we are easily convinced that the whole problem is not like one needing a carefully balanced equation.

Our whole concern always is—how likely are we to be able to satisfy the needs of about 1,200,000,000 tons of iron ore with the necessary quantity of manganiferous ore during the next 20 years. Beyond that time we may readily grant that many new conditions, requirements and equations will arise, because by that time we will be making a marked inroad into the “probable” ore reserves, about which greater speculation exists as to their character and worth than for the ores we are considering today.

There is no justification for believing that the ratio will be less than it is now; therefore, we must have no less than 48,000,000 tons. Our past performances show that the ratio has been increasing; so we must

arrive at some figure greater than 4 per cent. for the future period. Were we to extend the *A* line in Fig. 3 to intercept the vertical coordinate representing the year 1935, the ratio would be 2.9 per cent. If Lake's curve of future production⁸ is anywhere near correct, or is of value, it shows that in 1935 the total output of ore ought to be 65,000,000 tons. At 2.9 per cent., we would be producing only 1,900,000 tons of manganiferous ore, or less than we are using today.

Using the *C* line, the ratio indicated for 1935 is 5.4 per cent. By proportion, we find that if 4.2 per cent. in 1926 is applied to an output of 60,000,000 tons, then 5.4 per cent. would mean an output of 77,500,000 tons of iron ore in 1935. That is also an illogical result.

Using the *B* line, the ratio indicated for 1935 is 4.6 per cent. By proportion, as above, 4.6 per cent. would mean a total output of 65,700,000 tons. The closeness of that figure to Lake's figure for that year may be a coincidence, but it encourages extending the calculation for further test. Applying again the *B* line, but this time to the year 1946, a period 20 years hence, the plat scales 5.3 per cent. By proportion, we calculate a total output of 75,700,000 tons, which also compares favorably with Lake's figure of about 77,000,000 tons for 1946.

This gives us good reason for using 4.2 and 5.3 per cent. as the two extremes, for 1926 and 1946, respectively, from which we arrive at an average of 4.75 per cent. for the whole period, and that means a grand total requirement of manganiferous ore amounting to 57,000,000 tons. It should be stated that even were the period extended to 27 years, as Lake proposed, the average percentage resulting therefrom would produce no more than 60,000,000 tons as the grand total. It is admitted that the past does not determine the future events according to any mathematical formula; in this instance it is offered as the best basis or best corroborating evidence that is available.

When it comes to applying similar methods to the probable reserves, we are confronted with a vastly different calculation. Just as Lake has included many very speculative tonnages in his probable ore (but it is not the purpose here to analyze or even question those figures), so we may propose now that the trade may then be compelled to use more manganiferous ore with less than 4 per cent. manganese as now supplied abundantly by Michigan districts, and less than 8 to 10 per cent. manganese as now supplied by the Cuyuna district; or, the Cuyuna district may be called upon to supply 4 per cent. ore, as the Michigan districts have been doing for a quarter-century. If that should come to pass, the annual tonnage requirement might be trebled or quadrupled to supply the desired manganese units. Those conditions would then make it proper to include in the reserve what is now called manganiferous formation and is presently excluded from consideration, wherefore the larger

⁸ M. C. Lake: *Op. cit.*, 3.

requirement presents no great cause for worry today. Present estimates would look absurd if such allowances or inclusions were made now in estimating on such a basis. It is outside the purpose of this paper to go further into that portion of that study. It is merely mentioned to show what lies before us.

Available Estimates of Reserves

Minnesota Tax Commission.—In Minnesota all exploration data are submitted to the Commission for examination. The Minnesota School of Mines makes the estimates and the Commission fixes a valuation. The Commission publishes reports of all its work and summarizes the iron ore valuations. The tonnages reported vary from time to time according as mining or economic conditions disclose more or less taxable ore. Thirteen years ago the Commission reported approximately 10,000,000 tons of manganiferous ore for the Cuyuna district, whereas the same properties, without many significant changes due to mine work or drilling, are now reported containing about three times that quantity.

These estimates, it is important to note, are for ores that characterize Cuyuna manganiferous ores, say of 5 to 12 per cent. manganese content. Were ores of lower content included, the tonnage would be much greater. So far as is discernible, the records indicate that the total tonnage with taxable value is about 30,000,000. If smaller deposits were of value today and included, even this figure would be appreciably increased.

Mining Directory.—The School of Mines of the University of Minnesota publishes each year a booklet entitled "Mining Directory of Minnesota." It is a useful compendium, and it is often quoted. It lists every mine, and many properties that have been drilled and may some day become mines. For each one, a tonnage, available as of May 1 of the last taxable year, is printed. Those tonnages are estimates made by the School of Mines and used by the Tax Commission. They are not separated into iron ore and manganiferous ore. For anyone else to apply those figures in a summary of manganiferous ore tonnages, requires an intimate acquaintance with each property to enable him to make the necessary separations.

Bureau of Mines Estimates.—The Bureau has published tonnages;⁹ it does not estimate deposits, but compiles data others develop. Its recent news bulletin, citing tonnages for the Cuyuna district, is such a compilation, the figures being modified according to personal opinions obtained from local parties. The bulletin is quoted here, in part:

. . . based on the records of the School of Mines indicates that there are about 3,500,000 tons of black ore and about 24,000,000 tons of brown ore. The figures represent what might be called proven ore and do not indicate the ultimate tonnage which may be produced . . . In 1913 the Minnesota Tax Commission estimated

⁹ Bur. Mines News Bull. (Oct. 12, 1926).

the reserves of manganiferous ore in the Cuyuna district as approximately 10,000,000 tons. Information from the same source indicates that there are between 27,000,000 and 28,000,000 tons of manganiferous iron ore.

To one familiar with the local situation, this statement shows that the tonnage estimates for those ores are subject to processes of shrinkage and expansion, to suit one's needs; and it indicates that two kinds of manganiferous ores are involved, which is well known but may be emphasized because it bears on what has been suggested herein about mixing ores.

Michigan Tax Commission Reports.—The Michigan commission does not publish estimates or valuations in such detailed form as does the Minnesota commission. There is an additional difficulty for every estimator in obtaining data about the location and extent of orebodies now being mined to enable the estimators for the commission to provide as complete estimates for Michigan deposits as may be accomplished for most Minnesota deposits, as pointed out by Barrett.¹⁰ For Michigan, separate tonnages of manganiferous ores are not kept for purposes of taxation.

Available at this time of writing are the totals as of Jan. 1, 1925, which are, according to Barrett:

	Tons
Gogebic.....	52,026,380*
Marquette.....	68,987,647
Menominee.....	67,184,024
 Total.....	 188,198,051

* Michigan only.

Barrett, in a personal communication, states that from 1911 to 1924 discoveries have kept pace with shipments, and that since 1924 there has been a shrinkage, but that the reserves continue to hover around 190,000,000 and 200,000,000 tons. The average estimated reserve during the past 10 years for the whole Gogebic and Menominee ranges has been about 60,000,000 and 67,000,000 tons, respectively. These are not impressive totals when viewed by themselves; and when one considers that a part of them are manganiferous ore, the latter portion may not appear as an enticing quantity. The estimates will be analyzed subsequently.

Technical Societies.—The American Institute of Mining and Metallurgical Engineers and the Mining and Metallurgical Society of America jointly published a bulletin in 1925, entitled "International Control of Minerals." Pages 51 to 86, inclusive, are devoted to manganese. We need interest ourselves in only that part of the report pertaining to the ores of lower-manganese content. The committee received reports from all possible sources and examined some areas; it obtained records, to quote

¹⁰ L. P. Barrett: *Op. cit.*, 196.

its words, "of 1850 deposits, alleged deposits, and prospects," for the whole United States. In its summation of tonnages it divides the ores into classes, the one class containing ore with manganese contents ranging from 5 to 35 per cent. This would suggest that no deposits averaging less than 5 per cent. were used. In its Table D-II, page 71, Michigan is credited with 1,030,000 to 1,500,000 tons, and Minnesota with 26,000,000 to 35,000,000 tons.

Discussion of the Estimates

Minnesota Reserves.—The estimate of the Minnesota Tax Commission is unquestionably a reliable estimate of the principal deposits containing ores of the kind shipped to date, and may be considered a minimum of assured tonnage of that sort. The figures appearing in the Mining Directory and those quoted by the Bureau of Mines are but variations of those of the Tax Commission and offer no new data. It is quite certain that none of the tonnages listed includes manganiferous ore for the Mesabi district; consequently the tonnages apply only to the Cuyuna district. For 20 years the Mesabi has produced an annual average tonnage of 221,236 tons, and although averaging no more than 2.86 per cent. in manganese, that ore was used for its manganese, and it is included in the total annual requirement of 4 per cent., set forth in preceding pages. Using the Directory figures for the Cuyuna properties listed therein known to have only manganiferous ore and adding thereto nominal tonnages for other listed properties for which the estimate is not divided into iron ore and manganiferous ore, at least 33,000,000 tons can readily be compiled. At the present rate of production in the Cuyuna district, this would last about 30 years.

Many may disapprove considering ore as low as 2 per cent. in manganese; but inasmuch as a separate record has been kept of that kind for 25 years, and as the Gogebic district produced 141,259 tons for as recent a year as 1921 averaging only 2.81 per cent., it is certain that these ores have a well-defined use. Ores such as the Mesabi has produced are still very abundant there. The demand for them has been waning, perhaps because more desirable ores have become available elsewhere. It may even happen that they will not be listed again in annual records for many years, but they might then be absorbed right at the mine in local ore mixtures; the iron ores may be "graded" at the mine, as that term is used, for a higher manganese content than usual. That would help keep down the annual requirement of manganese units otherwise supplied by ores obtained elsewhere. Inasmuch as the Mesabi has in the past averaged an annual output of 221,236 tons, we may assume that it will be called upon to produce only 100,000 tons per year; for 20 years that would mean 2,000,000 tons. For the Mesabi, this is a very small figure,

as tonnages go. No one using that amount would be charged with overestimating.

Michigan Reserves.—For Michigan we find an odd situation. The records show (Table 2) that the Gogebic ore rarely exceeded 4 per cent. in manganese, and since 1917 the Menominee average has been 5.83 per cent. In 1926 both districts produced a total of about 1,116,662 tons. For the Gogebic the production for the last 10 years was 9.6 per cent. of all ores produced, and for the Menominee it was 5.8 per cent. At the present rate of production the estimated Michigan reserves of iron ore, unless augmented, would not last more than 12 years; but, as already stated, Barrett believes enough ore will be added from time to time to extend the life to perhaps 40 years. This opinion is not challenged herein; the figures are used as a basis of arriving at some estimate of tonnage of manganiferous ore.

The average annual output of manganiferous ore for the last 10 years is 9.6 and 5.8 per cent., respectively, of each district total for the Gogebic and Menominee districts. Would it be greatly in error to apply these same percentages to the total estimated reserve for each district? For the Gogebic that would figure 6,240,000 tons and for the Menominee 3,770,000 tons, or a total for Michigan of 10,010,000 tons. For the past 4 years the percentages were 14.3 and 6.5 per cent., respectively, and they would figure a reserve of 13,520,000 tons. This makes no allowance for beneficiation of poorer ore, and assumes that the ores occur in the ground in the same proportions as they have been mined. If we are willing to go on record that both districts will annually augment their reserves by new finds of ore in presently known properties, there is no valid reason why the same process should not apply to a manganiferous variety of ore. During the past 4 years the average production of manganiferous ore for the Gogebic district was 743,088 tons and for the Menominee 299,156 tons; if this continued for only 20 years, the total reserve today of manganiferous ore in Michigan (and Wisconsin) would be 20,844,880 tons.

Cuyuna Reserves.—The Cuyuna district is manifestly the bulwark of the whole problem. Here is a new district, having shipped only a few years, with a production of manganiferous ore already up to 1,000,000 tons, now constituting over one-half of the district annual output, and constituting one-half of all such ores produced annually in the region. The ores produced have averaged twice as much in manganese as any other produced, and the big bulk of material containing 15 to 25 per cent. manganese is not even being mined, nor is it ready for extraction. It has been demonstrated in an existing operation that mixing various non-usable ores can be done successfully, to dilute or increase various ingredients, and result in a very desirable ore. Drilling has been incomplete on most properties, some of which are even now estimated for taxation, and many nearby properties are unexplored. Some manganiferous ores, like

those mined in other districts, have not been considered in estimates cited. Truly, the possibilities are large.

The characteristics of the ores that have been used have already been described elsewhere¹¹ in detail by the writer. A black ore and a brown ore, and another kind high in manganese and silica and likely to be absorbed or improved in some manner, were described. The black ores are higher in manganese and silica and lower in phosphorus; the brown ores are more abundant, at present more cheaply produced, and more readily produced in large quantities. The existing mines are not being called upon to produce ore up to their operating capacities.

It is impossible for any one man to be in possession of the details for each and every property, now fully or partly developed or explored; and it is beyond the ability of any one man to evaluate the many uses or applications that will be made of the various kinds of ores, grades of ore, and the abundant manganese-rich formation already disclosed. It is with many apologies, and with a full appreciation of the shortcomings with which he may be charged, that the writer has endeavored to compile some tonnage figures of his own. He has used the estimates of others only as a guide in some special instances, wherefore he shares with no one the responsibility for the adequacy of the summation and accepts alone the adverse criticism it may create. First an estimate was made of the quantities of usable ore, and this was then increased for reasonable and likely extensions. It includes the tonnages of smaller deposits, because these may be extended. No illusionary possibilities were converted into figures. We may set down the Cuyuna district as possessing a reasonably safe reserve of 44,000,000 tons.

Now, if one will allow for the extension of the practice of mixing various manganiferous ores not usable by themselves, and adding thereto certain iron ores, as has already been done, who would venture a guess as to the additional tonnage thus created? If one will allow for the tremendous quantity of high-manganese formation, and which is, in fact, being experimented upon now with much success, the additional tonnage will treble or quadruple any estimate that anybody will offer today as known ore.

Summary

Five combinations, in which the various tonnage estimates just explained are used, are set forth as a summary of the available reserves. This offers a ready means of comparison, and shows impressively where the most favorable opportunities lie for meeting our demands.

¹¹ Carl Zapffe: Manganiferous Iron Ores of Cuyuna District, Minnesota. *Trans.* (1925) 71, 372.

Combination 1.—Available Reserves Manganiferous Iron Ores

[EMPLOYING: a, THE 1925 FIGURES BASED ON ESTIMATES BY STATE COMMISSIONS; b, THE AVERAGE PERCENTAGE FOR THE PAST 10 YEARS OF MANGANESE PRODUCTION.]

	ESTIMATED TONNAGE OF ALL GRADES 1925		MANGANESE ORE PRODUCTION LAST 10 YEARS, AVERAGE PER CENT.		CALCULATED RESERVE, TONS
Mesabi.....	1,253,442,107	×	0.8	=	10,027,537
Cuyuna.....	51,660,746	×	34.8	=	17,977,940
Gogebic.....	65,000,000	×	9.6	=	6,240,000
Menominee.....	67,184,024	×	5.8	=	4,996,673
Total.....					39,242,150

Combination 1 is clearly wrong because the reserve shown for the Cuyuna district by the method used is only about one-half of the estimate of such ore used by the State Commission as taxable ore.

Combination 2.—Available Reserves Manganiferous Iron Ores

[EMPLOYING: a, SAME TAXABLE TONNAGES AS IN COMBINATION 1; b, THE AVERAGE PERCENTAGE FOR PAST 4 YEARS OF MANGANESE PRODUCTION.]

	ESTIMATED TONNAGE OF ALL GRADES 1925		MANGANESE ORE PRODUCTION LAST 4 YEARS, AVERAGE PER CENT.		CALCULATED RESERVE, TONS
Mesabi.....	1,253,442,107	×	0.25	=	3,133,805
Cuyuna.....	51,660,746	×	50.00	=	25,830,372
Gogebic.....	65,000,000	×	14.3	=	9,295,000
Menominee.....	67,184,024	×	6.5	=	4,366,961
Total.....					42,626,138

[Combination 2, it may be noted, is based on the requirements of manganiferous ore experienced during the more recent years. It shows a marked reduction for the Mesabi reserve, and the Cuyuna reserve is still not as large as actual estimates produce.

Combination 3.—Available Reserves Manganiferous Iron Ores

[EMPLOYING FOR MICHIGAN AVERAGE PRODUCTION OF PAST 4 YEARS AND ASSUMING IT TO REPRESENT AVERAGE PRODUCTION FOR 20 YEARS, OR ONE-HALF PERIOD OF LIFE PREDICATED BY L. P. BARRETT.]

		CALCULATED RESERVE, TONS
Mesabi (Using a nominal quantity, see page 363).....		2,000,000
Cuyuna (Using approximate total of Minnesota Tax Commission).....		30,000,000
Gogebic.....	743,088 tons, average of past 4 years.	
Menominee.....	299,156 tons, average of past 4 years.	
	1,042,244 tons, average of past 4 years.	
	× 20 years.....	20,844,880
Total.....		52,844,880

This combination more than satisfies a minimum requirement of 48,000,000 tons.

Combination 4.—Available Reserves Manganiferous Iron Ores

[USING THE SPECIAL CUYUNA ESTIMATE.]

	CALCULATED RESERVE, TONS
Mesabi (See Combination 3).....	2,000,000
Cuyuna (See page 365).....	44,000,000
Gogebic and Menominee (See Combination 3).....	20,844,880
Total.....	66,844,880

This combination satisfies the demand of 60,000,000 tons which would arise to cover the estimated total requirement up to 1946 (see page 360).

Combination 5.—Available Reserves Manganiferous Iron Ores

[USING THE LARGEST ESTIMATE FOR EACH DISTRICT.]

	CALCULATED RESERVE, TONS
Mesabi (See Combination 1).....	10,027,537
Cuyuna (See Combination 4).....	44,000,000
Gogebic and Menominee (See Combination 3, but using 25 years) ..	26,056,100
Total.....	80,283,637
Cuyuna, additional quantity resulting from beneficiation or mixing.	Not Calculable today
Grand Total.....	(?),000,000

It is of interest to record one more statement made by Barrett.¹² He shows by a series of curves what the life of the ore deposits for various Michigan counties are likely to be and the tonnages they would, therefore, produce; or what those counties could be assumed to contain today although not yet definitely disclosed. For Iron County, and for unexplored ranges which he does not designate, he arrives at a total of 456,000,000 tons. This he fears is too large. Since all the manganiferous ores of the Menominee district are mined only in this county, and as it constitutes today 5.8 to 6.5 per cent. of its total annual production, we would have, therefore, a reserve of about 27,000,000 tons in this county. It happens that nearly all the manganiferous ore produced in Michigan comes from Iron County. With Gogebic County (and Wisconsin) figured in the same manner, these two mining districts should easily contribute 40,000,000 tons. Others with whom the writer has conferred on this subject, and who are well qualified to pass thereon, hold views as to future production very much in accord with this latter figure. Consequently, using only 20,000,000 to 26,000,000 tons, as was done in Combinations

¹² L. P. Barrett: *Op. Cit.* 197, 198.

3, 4 and 5, indicates that there is a considerable degree of safety in the method of calculation employed.

CONCLUSIONS

An attempt has been made herein to show (1) the existing data pertaining to manganiferous iron ores and to explain the existing estimates thereof, and (2) the generally accepted data pertaining to iron ore reserves, their consumption, and their expected life. So far as it has been possible, an interpolation has been made of the likely tonnages and grades where estimates of deposits of manganiferous ore did not exist. It was shown that the likely requirement of manganiferous iron ore needed to satisfy the known reserves of iron ore in the Lake Superior region could hardly be less than 48,000,000 tons, and that at the outside, based on recognized practices of today, no greater total requirement than 60,000,000 tons can be figured for the next 27 years.

The study has disclosed that our manganiferous reserves can not now be definitely stated in tons because the estimators of the various districts do not always differentiate in their estimates between iron ore and manganiferous ore along division lines as determined by the specific employment of those ores; nor are they fully explored. By using, however, the least speculative factors based on usage, it is found that the requirement of 48,000,000 tons can easily be met, when the estimate of one district alone virtually shows that quantity now.

The study further discloses a great lack of exploration on properties known to contain such ores, and on adjoining unexplored property. But by assuming that the same kinds of ores as have been used for 25 years will be the prevailing kind for many years to come, we can proceed with confidence that the 60,000,000-ton requirement, based on present estimates of assured iron ore reserves, will be satisfied. And if we will permit ourselves to place faith in our own abilities to extend our practices of beneficiation, as we have already done with iron ores when that became imperative, and to devise new and better methods of using the manganiferous formation that nature has supplied us with in abundance, we may look with confidence toward the days when we attack the tonnage of probable iron-ore reserves.

APPENDIX

The author has made several test calculations on the basis of manganese units that are involved. These are summarized here, as follows:

On the basis of ore produced since 1902, the average annual requirement of manganese units would be 91,000 tons; on the basis of the production of the past 4 or 5 years, the requirement would be 145,000 tons; on the basis of the year 1925 only, the requirement would be 155,000

tons. Applying the percentage of increase for the next 20 years, as determined by the plat shown as Fig. 3, the total requirement for 20 years would be 3,254,000 to 3,572,000 tons, if one used either 145,000 or 155,000 tons as a starting point.

We have the following interesting facts: (1) The Michigan ores contain only about one-half the manganese that the Cuyuna ore does; (2) the Cuyuna district now produces one-half the ore required; (3) the known tonnage of Cuyuna reserves is greater than the speculative reserve offered for Michigan; (4) at the present rate of production considered by tons, Michigan ores ought to be exhausted in 20 years or less while Cuyuna ores would last up to 44 years.

We should now reduce the estimated reserves to a basis of manganese units. Using the tonnages listed in Combinations 2, 3, 4 and 5, the reserves total 3,118,228 tons, 3,781,995 tons, 5,041,995 tons and 5,560,342 tons, respectively. Therefore, needing at the most 3,572,000 tons, as shown above, either Combination 3, 4 or 5 will not only suffice but will leave an excess—enough for 1 or 2 years for Combination 3, 9 or 10 years for Combination 4, and 12 to 13 years for Combination 5. This means that for Combination 5, without using tonnages obtained from beneficiating or from mixing poor ores, the reserves are ample for about 33 years.

If this calculation is extended to disclose the life of each district, based on manganese units delivered each year and units in reserve, the average content delivered in just the past year or two being maintained and taken as a representative average, it would seem that the Gogebic district would not be producing ore 20 years hence. The situation in the Menominee is more doubtful; it may or may not be producing. The Mesabi district will unquestionably produce ore for many years in excess of 20, even if ore with a content of 2 to 3 per cent. manganese should find a greatly increased application. The Cuyuna district has a reserve tonnage of ore of 44,000,000 tons. If it averages 9 per cent. in manganese, as the ore produced at present, it has a reserve of 3,960,000 units. If it produces 1,000,000 tons of ore per year, as now, averaging 9 per cent. in manganese, in 20 years it will contribute 1,800,000 units, and there would still be available 2,160,000 units. The Michigan districts should then be exhausted, as seems possible. Because the industry would need in 1946, about 175,000 units per year, it is apparent that the Cuyuna might become the sole producer and be such for the following 12 years at least. Therefore the industry is assured a supply for at least 32 years, and the Mesabi could be reckoned to help extend this period.

DISCUSSION

L. P. BARRETT, Lansing, Mich. (written discussion).—On January 1 of this year, it was decided that a separate estimate would be made of all Michigan ores carrying over 2.5 per cent. manganese, dried analysis. The reserve included in Table 3 were

estimated before I received the copy of Mr. Zapffe's paper and without knowledge of his methods. They were based on customary methods of estimating ore reserves on underground properties and drill reserves from data furnished by the mining companies to the Michigan Appraiser of Mines.

The summarized estimate of manganiferous iron ore in Table 3 is therefore from deposits that will be included as taxable reserves for the year 1927. These ores are classified into three main grades; those carrying between 2.5 and 4 per cent. manganese, between 4 and 10 per cent., and over 10 per cent. The analyses are dried, as it was not found possible to calculate the reserves to natural iron on account of the absence of data on the moisture content from analyses furnished with the mine maps and drill records. The reserves are also classified into three groups; developed, prospective, and potential. The developed reserves require no explanation. Prospective reserves are those additions to developed tonnage which will be used for valuation of the properties for purposes of state taxation for the year 1927. The potential reserves in this particular estimate represented the extensions to known developed deposits which, while possible of realization, are not sufficiently certain to warrant their inclusion as a basis for taxation.

The term "potential" as used in this table does not include tonnage added for possible new discoveries within the districts as a whole. The total of 11,164,000 tons may be treated as a reasonably assured reserve.

TABLE 3.—*Reserves of Manganese Iron Ore from the Marquette, Gogebic and Menominee Iron Ranges*

	Devel- oped	Prospe- ctive	Poten- tial	Total	Iron	Manga- nese	Phos- phor- us	Sul- fur	Silica
Manganese under 4 per cent.....	6,257,569	955,506	700,000	7,913,075	51.80	3.66	0.212	0.038	4.92
Manganese between 4 and 10 per cent.	430,912	450,000	700,000	1,580,912	54.0	6.37	0.225		6.62
Manganese over 10 per cent.....	813,685	57,256	800,000	1,670,941	39.60	12.50	0.603		4.83
	7,502,166	1,462,762	2,200,000	11,164,928	50.30	5.37	0.263		5.14

The manganiferous reserves outlined in the above table represent the total for all ranges of Michigan. I regret the fact that it is impossible for me to make public the division of the tonnage between the various districts, but this cannot be done without in some measure revealing the individual ore reserves, which is contrary to the established policy of the Michigan State Geological Survey and the Michigan State Tax Department to treat individual mine tonnages as confidential information.

Michigan iron ores are derived from two principal geological formations, one of Upper and the other of Middle Huronian age. The high-phosphorus ores of Iron County and the low-grade ore from Ford's Imperial mine are of Upper Huronian derivation. All of the remaining production of Michigan comes from Middle Huronian iron formation. With the exception of the Middle Huronian iron formation in Dickinson County, all of the iron-bearing districts in Michigan have produced some manganiferous grades and at the present time deposits of iron ore carrying various percentages of manganese are known in all of the districts. Recently drilling in the Marquette district has added small reserves of manganiferous ore from that range. I note that Mr. Zapffe makes no mention of the Marquette district as a possible source of manganiferous ore. While it is true that the production of manganese on the Marquette Range is inconsequential, the fact of the occurrence of small amounts

of manganese with the iron ore in that district is well established and recently a small reserve has been proved by diamond drilling. There is an old prospect near the Breitung No. 2 mine, which was called "The Manganese" and which shipped a few thousand tons of manganese ores. Small local concentrations of manganiferous ores are known to exist in several of the mines in this range although not in sufficient quantity to permit segregation and extraction as a manganiferous grade. The occurrence of the element manganese, however, even though in too small amounts to warrant mining of separate grades, is sufficient to justify the belief that future exploration on this range may result in discovery of some manganiferous ores. On the Gogebic Range, manganiferous ores are found in several localities, but the geologically important one is the Plymouth or Lower member of iron-bearing formation. In Iron County, manganiferous ores are found rather widely distributed throughout the iron formation. The present reserves of manganiferous ores listed in Table 3 constitute about 4.7 per cent. of the total assured reserves of iron ore. It is probably a safe and reasonable assumption that additional discoveries in the Michigan iron districts will maintain this ratio.

C. P. McCORMACK, Cleveland, Ohio.—Manganiferous ore is used as a medicine in making 2 per cent. manganese pig iron, therefore, there should be an even grade of manganese. The biggest problem of Michigan and Minnesota is to ship an even grade. To obtain a uniform manganese means that the ore must be crushed and mixed very thoroughly, which adds appreciably to the cost, and if the use of manganiferous ores from the Lake Superior region is to be a wide practice, a higher value must be obtained for the manganese than is being obtained today.

Manganese Ore Deposits of the Gold Coast, Africa

BY SIR ALBERT E. KITSON,* LONDON, ENGLAND.

WITH NOTES ON THE PETROLOGY OF CERTAIN ASSOCIATED
MANGANESE SILICATE-BEARING ROCKS

BY MAJOR N. R. JUNNER,† M. C., LONDON, ENGLAND

(Cleveland Meeting, April, 1927)

GENERAL GEOLOGIC RELATIONS

THE manganese ore deposits of the Gold Coast, British West Africa, occur in very ancient rocks, of both sedimentary and metamorphic types. In certain respects they have a strong resemblance to those of India and Brazil, as contrasted with those of Tertiary age in Georgia, but in this paper no careful comparison with them can be made.

Viewed from the present condition of the manganiferous rocks, the oldest deposits are in the sediments. Later than these are the progressively metamorphosed representatives of these sediments. Youngest are the products of decay and degradation with the concentration of manganese oxide in the rocks of both groups.

The oldest occurrences are in argillaceous and siliceous phyllites and slates, which show only a moderate degree of metamorphism. They belong to a series which also includes mudstones, sandstones, grits, conglomerates and subordinate limestones, with tuffs and lava-flows, largely of basic types. This has been named the Birrim System. Its age is unknown, for no definite fossils have been discovered in it. Since these non-fossiliferous strata at many places are of a character which in many parts of the world are suitable for the preservation of fossils, and which contain them, it is assumed that the Gold Coast deposits are pre-Cambrian.

These rocks of the Birrim System have normally a northeast-south-west strike, and are folded, but not to any very great degree except in certain belts. From the Eastern Province of the Gold Coast they may be traversed diagonally across their strike for upwards of 600 miles, to the extreme northwestern corner of the Northern Territories. Thence they probably extend much farther into French territory.

* Director, Geological Survey, Gold Coast, British West Africa.

† Director, Geological Survey, Sierra Leone, British West Africa, formerly Geologist, Gold Coast Geological Survey.

Intrusive into these sediments are great masses of granites of two distinct periods; an older one—medium to dark gray—essentially a biotite-granite, and a younger one of much lighter color, a muscovite-granite, or biotite-muscovite-granite.

These granitic intrusions especially have caused a great deal of contact-metamorphism, and the formation of crystalline schists and para-gneisses. They are normally strike-intrusions. But in other districts these sediments have been only partly altered, as if by underlying batholiths, and the metamorphism has not advanced beyond the stage of fine schists and hornfels.

Intrusions of a still younger type of acidic rock, varying from a granodiorite to a porphyry, occur over a large portion of the country. There are also other smaller intrusions of intermediate, basic and ultra-basic rocks.

Imposed upon all there has been pressure-metamorphism on a grand scale in certain districts, and the sediments and intrusives alike have been completely changed into para- and ortho-gneisses and schists of various kinds. Thus certain beds consist of slightly altered rocks in some districts and in others occur as gneisses and schists along the strike of their beds, or in their folded lateral extensions.

It is to be expected, therefore, that in such a thick series, extending over a great width of country, any beds with a distinctive composition, such as manganiferous slates, will present entirely different features at different places, depending on the degree of metamorphism that they have undergone. Thus they may easily be regarded as belonging to entirely different periods. It was the consideration of this aspect of the question for correlation purposes that led the writer to attach much importance to manganiferous rocks, and to suspect strongly from the field evidence that the altered intrusive rocks of certain crystalline complexes are really younger than many of the comparatively unaltered sediments. The results of microscopical examination of these various types by Major Junner have helped to prove the younger age of many rocks generally regarded as Archaean. The original Birrimian sediments have a group of beds that contain varying proportions of manganese oxide in their composition. These beds are of both argillaceous and arenaceous character and were apparently deposited on a fairly deep sea bottom. With the mud and fine sand was deposited oxide of manganese—whether due to ordinary chemical action, or to bacterial, or to both, can be merely surmised. The least altered of these now occur as slates, phyllites and indurated sandstones, and contain manganese oxide intimately associated with them, the argillaceous ones having it in the greater proportion.

By progressive stages of metamorphism the manganese oxide has undergone the changes that might be expected; viz., the formation of

manganese-garnet (spessartite) from crystals of microscopic size in the less altered rock, still phyllites, fine schists and hornfels, to those larger than peas in the completely altered ones, garnetiferous para-gneisses and schists. Examples of these have been noted at a number of places, and in the appendix their microscopical character is described by Major Junner.

TYPES OF MANGANESE DEPOSITS

In neither the slightly altered rocks, nor in those completely crystallized, however, has manganese oxide been found in sufficient quantity to be of economic value. It is only in those that have been greatly affected by tropical climatic conditions that commercial deposits have been found.

Weathering agencies have operated strongly, and under their influence the originally deposited manganese oxide has been dissolved and reprecipitated in suitable places, thus forming by concentration at certain localities masses of material containing manganese oxide in sufficient proportions to make the deposit of economic value. There are two types of this concentration, massive and concretionary.

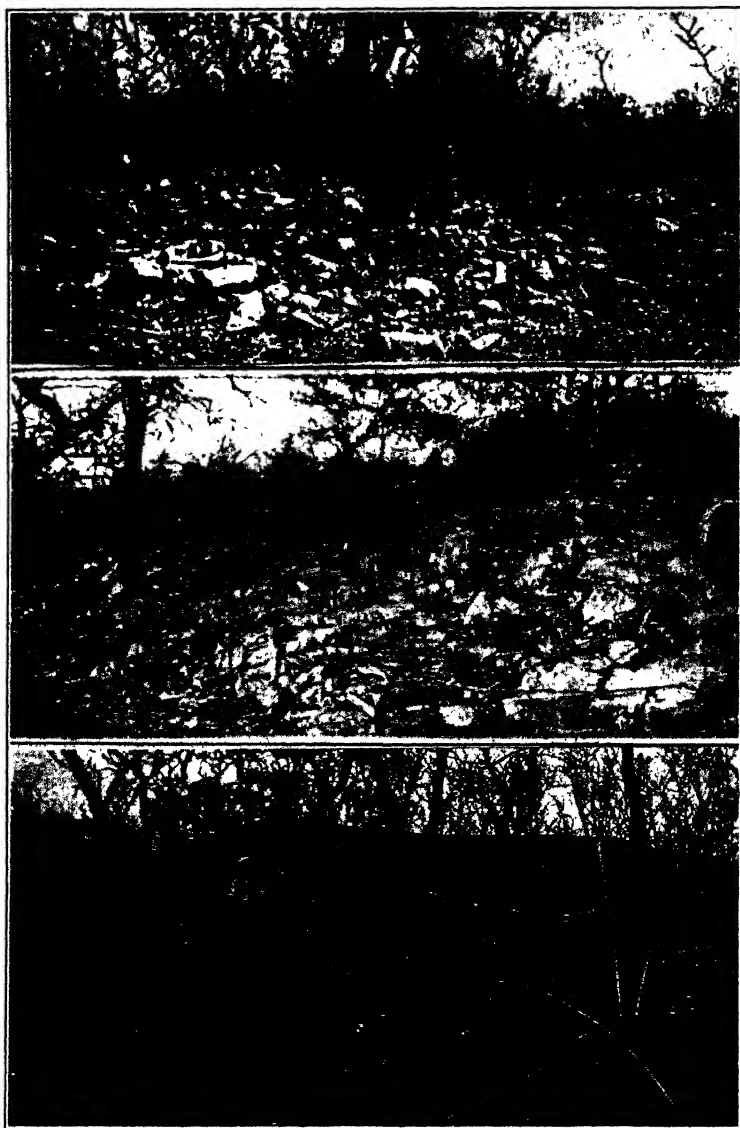
In the massive type there has been a steady addition of the oxide to the rock with which it was associated, but at lower depths in the same bed, or contiguous ones, with an accompanying removal of the argillaceous or siliceous material of the bed. Such action obviously depends on physical conditions of the rocks and percolation of water-bearing manganese oxide in solution. That this action is still in operation is proved by the rapid deposition of a film of manganese oxide on quartz or other rock over which the local water flows.

In the concretionary type no such physical conditions are necessary and the deposition of the oxide has taken place in the main mass of the disintegrated argillaceous rocks—the soil and subsoil—at those places where climatic conditions, such as a heavy rainfall with suitable natural aids, as a more or less thick cap of soil, allow slow percolation of water and where vegetation has protected the soil from being rapidly washed away by heavy tropical rain.

Thus one may expect economic deposits of manganese ore of massive type to be formed principally in rocks of argillaceous kinds. Similarly, though to a less degree, the formation of economic deposits of concretionary ore, for a sandy soil would be less likely to admit of deposition of the oxide and the quality of the deposit would be much more siliceous by reason of the highly siliceous and larger nuclei forming the cores of the granules of ore.

KNOWN MANGANESE DEPOSITS OF THE GOLD COAST

Fig. 1 shows the location of the manganese deposits of the Gold Coast. Those in the Gold Coast Colony are at, 1, Insuta-Dagwin, near Tarkwa;



3

FIG. 2.—MANGANIFEROUS SILICEOUS AND ARGILLACEOUS PHYLLITES, PONGO, SIRO-WA TRACK, NORTHERN TERRITORIES.

FIG. 3.—MANGANIFEROUS SILICEOUS PHYLLITES, KALIMBI HILL, NORTHERN TERRITORIES.

FIG. 4.—MANGANIFEROUS QUARTZ-VEINED SILICEOUS PHYLLITES, KALIMBI HILL, NORTHERN TERRITORIES.

2, Butre River, two localities; 3, North of Sekondi, two localities, near Adansi; and 4, east of Manso railway station, Sekondi district.

Deposits in Ashanti are at Odumase, near Konongo station, Accra-Kumasi railway, and at Birrinsim Hill, northwest of Bompata, Juaso district.

Deposits in the Northern Territories are at 1, Kalimbi Hill, southwest of Siripe, (Figs. 3 and 4); 2, Bole district, several occurrences; 3, between Wandara and Bongfu, 4, between Bole and Wandara; 5, at Pongo; 6, at and near Hapa, Lawra district; 7, Bele and Kundugi, Wa district; 8, at Tindongo, Zuaragu district; 9, near Pwalagu, Zuaragu district.

INSUTA-DAGWIN DEPOSIT

This deposit is immeasurably the largest and most important known in the country (Figs. 5 and 6). It was found by the writer on May 9, 1914, during the first tour of the Survey. The deposits have been traced definitely and almost continuously for about $2\frac{1}{2}$ miles northeast along a ridge from close to the Sekondi-Kumasi railway near the 34-mi. post

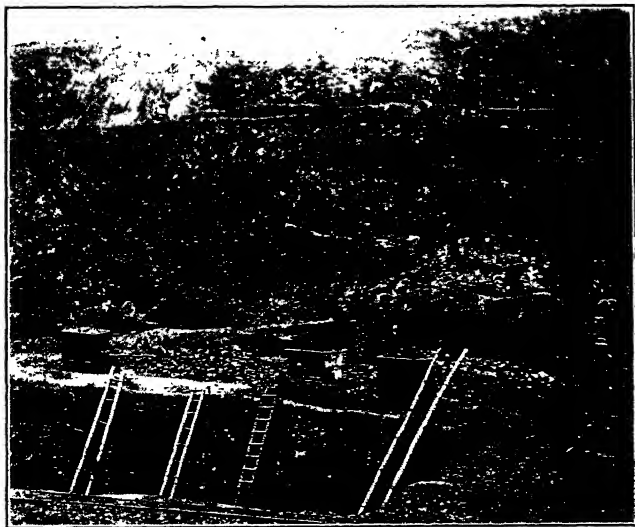


FIG. 5.—MASSIVE MANGANESE ORE, INSUTA MINE.

from the port of Sekondi. It is in a thickly forested belt with an average annual rainfall of about 70 inches.

At the southwestern end (Hill A) this ridge ends steeply close to the railway. The country thence southward is flat, and in part swampy, with no visible outcrop of rocks or manganese ore. To the northeast the ridge rises steadily in height above the valleys to the northwest and

southeast, and beyond Insuta railway station it is more than 400 ft. above the valley.

When the deposit was discovered, most of this ridge was covered with thick bush, among which great masses of ore were revealed as the vegetation was cut. No outcrops of rocks were visible, so the origin of the deposit could not be ascertained. It was not until an adit had been driven through the ridge at Insuta, about 150 ft. below the top, after the development of the deposit in 1916 had been commenced, that any evidence was found of the character of the underlying rocks. Some of this rock was reported to have been sent by the Fanti Consolidated Co., which then owned the mine, to the Imperial College of Science, London.

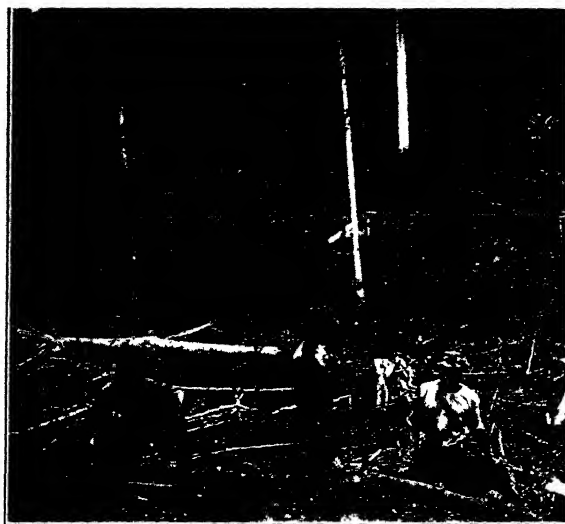


FIG. 6.—INSUTA-DAGWIN DEPOSIT. LOOKING TO 265° OVER SEKONDI-KUMASI RAILWAY FROM FIRST MANGANESE OUTCROP DISCOVERED BY GEOLOGICAL SURVEY, MAY 9, 1914.

where it was examined microscopically by Dr. Arthur Holmes and proved to be manganese-bearing phyllite.

No attempt can be made in this paper to give more than a very general description of the deposits and the mine, as for various reasons, no systematic examination of the occurrence has been made by the Geological Survey and pressure of official duties prevents more than a cursory description at the moment. The main features may, however, be given.

This ridge, really a double parallel one at the northeastern end, consists of a thick series of manganiferous phyllites, apparently dipping to the northwest at comparatively high angles. To the northwest it is flanked by the varyingly narrow and wide valley of the Kawere stream,

which now flows through what was a marine valley of erosion, in which are short, isolated and parallel ridges. The little evidence obtained shows that to the northwest there is a comparatively wide belt of argillaceous rocks, consisting probably of tuffs and lava-flows of the Birrim System. Farther to the northwest, across the strike, is a thick conglomerate, almost wholly of argillaceous character, consisting of decaying and decayed pebbles of basic volcanic rock. This has been called the Kawere conglomerate, and probably forms the basal bed of the thick series that contains the Tarkwa-Abosso goldfield. The principal feature of this series is the occurrence of auriferous conglomerates very strongly resembling the banket of the Witwatersrand. To this has been given the name of the Tarkwa System. It rests unconformably upon the Birrimian.

To the southeast of the Insuta ridge are other zones of rocks, sedimentary and igneous—some manganiferous. No work by the Geological Survey has been done there, so the character of the rocks to the southeast cannot be given.

There is a strong suggestion that dykes of porphyrite and porphyry, associated with the Dixcove type of intrusion of granodiorite, have cut through these phyllites and have also intruded along their strike. These rocks are visible farther to the south in railway sections. At one point in the mouth of an adit on Hill D, Insuta, the writer washed some of the soil at a point where in color and appearance it was much lighter than that of the general mass. This gave a considerable proportion of granular quartz showing no sign of attrition and resembling that obtainable from the soil of rocks of the grano-diorite type.

Decomposition of rocks on this ridge has been very great. The adit proves that even at 150 ft. below the surface in the heart of the ridge, the phyllites show such a considerable alteration that they are really enriched ones. At various places high-grade manganese ore shows very definite stratification, the replacement of the original material of the phyllite being beautiful and complete.

The surface of the ridge at Insuta consists of a mass of red clayey soil, which for more than 100 ft. in depth contains vast numbers of granules, nodules and pieces of manganese oxide. This so-called "detrital" ore is being excavated by steam shovels, washed by gravitation and tipped from hoppers into the railway trucks, 300 to 400 ft. below.

The proportion of granular and nodular manganese oxide of high grade in the soil is high, and shows the enormous amount of ore probably obtainable from the mine. In the report of the meeting of the Fanti Consolidated Co. on the operations for the year ended December 31, 1921, it is stated that by washing this material in a primitive plant, as was then being done, the company was getting from the mass 54 per cent. of manganese ore. With the improved plant now working the percentage of ore obtained has been raised to 75. A panning test

made by myself of some of this surface material from Hill E gave over 60 per cent. of good manganese ore.

In the report mentioned the estimate of the "detrital" ore is given as 3,000,000 tons. It may, however, be safely regarded that later developments have shown that this quantity can be very greatly increased.

In addition to this detrital ore, the work of development has shown the existence of a number of bodies of massive ore. The reason for their occurrence is not clear, and until work has progressed at deeper levels, and adits have displayed clearly the nature and disposition of the rocks, no satisfactory opinion can be formed.

Judging, however, by the disposition of these bodies of massive ore, they seem, in the main, to have a general north-south strike, and show parallelism with one another, crossing diagonally the strike of the sediments. It appears probable that these orebodies have been formed along lines of fracture, possibly with slight faulting, and that along these fractures there has been free circulation of meteoric water, with consequent solution of the original manganese oxide of the beds and its redeposition with replacement at lower levels.

From one of these occurrences of massive ore there is now issuing a small spring of water. Its course is visible on the slope of Hill D for some 50 yd. Along it big blocks of manganese ore can be seen. At another place on Hill D, an adit cut an orebody. This was driven along until trouble with water caused a suspension of operations, when another adit was made at a point some distance to the south of the former. This also cut an orebody, presumably the same one, for it also struck water, and drained the former adit. No further work has been done in either adit, owing to the necessity for retimbering or using steel sets.

The massive ore consists mainly of the ordinary type, with subordinate proportions of banded, vesicular, mammillary, stalactitic, and, rarely, acicular types. The last three occur in vugs from which the original material has been removed in solution. The process of removal of quartz is interesting, though not fully understood. It appears as though the normal white, opaque vein-quartz, of low-temperature origin, had been considerably crushed and later finely granulated before being removed. Lumps of ore, with quartz in various stages of alteration, from the compact to the finely saccharoidal kind, may occasionally be seen.

The principal mineral appears to be psilomelane, with less pyrolusite, and a little manganite and polianite. No rhodonite or rhodochrosite has been definitely determined in the least altered deposits, such as those at Insuta, though rhodonite is present in the deposits associated with the highly altered sediments—para-gneisses and para-schists—as, for instance, those near Yabio and Adansi.

MODE OF WORKING THE DEPOSIT

The ore is excavated chiefly by steam-shovels, of 1.5-yd. capacity, on caterpillars, and is loaded into cars of 4-cu. yd. capacity. These are dumped on to a 5-in. grizzly, the fines going through to a hopper, and the oversize to a jaw-crusher. The oversize is reduced to 5-in. size and returned to the hopper; from there it goes by apron-feeder to a large revolving screen. All material under 2 in. is fed at once to a launder, the coarser material going into a washing-barrel.

The launders lead to turnover-boxes, where the ore is sized to $-\frac{1}{4}$ in., $+\frac{1}{4}$ in., -2 in. and +2 in. The +2-in. ore is taken to another revolving screen, thence along a sorting belt into a bin, and its washing is finished. The -2-in. and $+\frac{1}{4}$ -in. ore goes to a 25-ft. log-washer and is delivered direct into a washed-ore bin. The sludge and $-\frac{1}{4}$ -in. ore from the log-washer, plus sludge and $-\frac{1}{4}$ -in. ore from the revolving screen, plus sludge and $-\frac{1}{4}$ -in. ore from the turnover-box, go to a settling tank. There the mud and sludge are taken off over the top, and the fine ore passed over an 18-ft. log-washer. From this it is fed on to a conveyor belt, and delivered to a washed-ore bin. The sludge from the settling tank and the 18-ft. log-washer is pumped by a centrifugal pump into a slime drain on the eastern side of the ridge.

The washed ore is taken by endless ropeway, in trucks of 1-cu. yd. capacity, on to a steel trestle that is 26 ft. high and 880 ft. long; it is dumped over the side, and loaded from there by steam-shovel (2-cu. yd. bucket) into railway trucks; then run over the company's weighbridge and despatched to Sekondi for shipment.

A stacking floor with belt-elevators is nearing completion. This will hold 250,000 to 300,000 tons of washed ore. It is intended to keep this quantity continuously in reserve.

The rolling-stock and track comprise 12 locomotives, 200 trucks of 4-cu. yd. capacity, and about 15 miles of 35-lb. track on steel sleepers.

The plant is able to produce 600,000 tons of washed-ore per year, but this quantity is not produced at present owing to the limited shipping facilities at Sekondi. The position will be changed entirely with the completion in 1928 of the construction of Takoradi harbor near Sekondi. The capacity of the washing plant is 50,000 tons per month, but the four shovels in use are able to supply 60,000 tons per month. The whole of the main mass of Hill D is at present being mined and put through the washing plant, and 75 per cent. by weight of the whole is marketable manganese ore. Of the massive ore, the percentage is above 95. The total quantity of ore exported from this mine up to March 31, 1926, was about 1,016,000 tons, valued at nearly £2,000,000. The mine employs about 50 Europeans and 2000 Africans.

BUTRE RIVER DEPOSITS

The Butre River deposits have not been examined by the Survey, so no description of them can be given. No work is being done on their development.

DEPOSITS NORTH OF SEKONDI

The deposits to the north of Sekondi, at Yobio in the Chama district, occur in highly metamorphosed sediments and basic rocks, and show small patches of secondary manganese oxide; but they are not of economic importance.

THE ADANSI DEPOSIT

No examination by the Survey has been made of the Adansi deposit, but a sample of some of the material from there, kindly given to the Survey by E. Payne, proved to be manganese-garnet, rhodonite and quartz, associated with para-gneiss and schist, so it is obviously in highly metamorphosed sediments.

ODUMASE DEPOSIT

This deposit, found by W. G. G. Cooper and the author in 1922, occurs at about 3 miles to the west of Konongo railway station on the Accra-Kumasi railway, which cuts through it nearly at right-angles to the strike of the beds. The village of Odumase is a short distance to the north of it. The rocks are hard dark gray to black manganiferous siliceous slates and flagstones, with a few thin argillaceous bands. The beds dip to 330° (north 30° West) at 45° to 65° . Red clayey soil, derived probably from phyllites, masks the rocks on the northwestern side. On the northeastern side the beds are in contact with clayey sericitic and sandy rocks with a little quartz. The visible thickness of the beds is about 38 ft. One part, some 25 ft. thick, has a considerable proportion of manganese oxide. Up the slope to the northeast are small outcrops and slabs of these rocks, but on the plain above an odd fragment of slate and small nodules of good manganese ore are the only indications of its existence.

Along the strike to the southwest a large outcrop occurs on the top of a hill less than 1 mile from the railway. The rocks there have the same general appearance, strike and dip, as in the railway cutting. Fragments of the manganiferous rocks are visible between this hill and the railway.

A sample, *A*, collected at intervals of 18 in. across the 25 ft. thickness of beds in the railway cutting, and another, *B*, at intervals across the outcrop on the hill to the southwest, analyzed as follows: *A*, Mn, 17.2 per cent.; Fe, 5.8 per cent.; P_2O_5 , 0.05 per cent.; and *B*, Mn, 16.0 per cent.; Fe, 3.8 per cent.; P_2O_5 , 0.10 per cent.

A survey of this belt of country was commenced by Dr. Teale in December, 1924, but it was suspended almost immediately because of his return to England. There has been no opportunity since to renew it.

Seven samples of the rocks from these beds, over a width of $35\frac{1}{2}$ ft., were collected and analyzed for manganese and iron by Dr. Teale. The analyses showed the manganese to vary between 17.08 and 24.54 per cent., and the iron between 4.0 and 4.8 per cent., averaging 22.22 per cent. manganese and 4.37 per cent. iron.

These results show that the material is valueless as an ore of manganese, and the almost entire absence of secondary ore, either from the beds or the soil, proves that this deposit is of no present commercial value.

BIRRINSIM HILL DEPOSIT

This hill is about 7 miles northeast of Konongo and apparently on the same line as that of Odumase. The rocks comprise decaying purplish blue sericite, chlorite, and quartz-schists, dipping to 80° and east at 45° to 75° . They show a considerable degree of staining by manganese oxide and a little secondary pyrolusite or psilomelane. At the northern foot of the hill, about 300 ft. below the top, there is a small amount of secondary ore in the form of granules and nodules in red, clayey soil, a mode of occurrence similar to that at Insuta.

No examination has been made of the country between this hill and the Odumase deposit, so it is not known if the two deposits are in the same group of beds. This, however, is probable. The deposit was found by Dr. Teale in 1918.

KALIMBI HILL DEPOSIT

The narrow, isolated Kalimbi hill stands sharply above the adjacent plain, about 250 ft. above it at the northern end. Its higher part consists of dark to medium gray, highly siliceous phyllites, containing great numbers of veins and veinlets of quartz. The rocks are highly contorted at some places, but the general strike is nearly north-south. There has been a little concentration of manganese oxide here and there on the higher parts of the hill; on the lower portions to the south southwest it occurs in larger quantity. It is, however, not of economic value.

The deposit occurs in a narrow belt of partly metamorphosed sediments, lying in a wide area of para-gneisses, ortho-gneisses and granite, and was discovered by O. A. L. Whitelaw in 1924.

BOLE DISTRICT

All the other occurrences found in this district in 1924 by Dr. Teale and Mr. Whitelaw are in the more highly metamorphosed sediments. The deposits are valueless, as there is little concentrated manganese oxide. The same may be said of those in the Wandara district, found by Major Junner.

PONGO DEPOSIT

This deposit, found by the author in 1924, occurs on a low V-shaped ridge. In its western portion the rocks are fine and coarse mica-schists, with strikes of 350° to 5° , and dips to 260° to 275° at 50° to 65° . There is very little visible evidence of concentration of manganese oxide.

In the other portion the rocks are siliceous, with strikes of 40° to 60° and dips generally to southeast. Several large masses of rock on the point of this ridge show quite a considerable concentration of manganese oxide. The deposit is economically valueless.

HAPA DEPOSITS

The deposits in the Hapa locality are visible on three hills: Zinzo, close to Hapa; Pabini, about 1 mile to south southwest of Hapa; and Pabini Bie, a short distance to east southeast of Pabini. They were discovered by F. Oates and the author in 1924.

The rocks on the first two are argillaceous and siliceous slates and phyllites, mauve, gray and greenish in color. They dip to east southeast at high angles on one point of Pabini, and to east northeast at 40° on the other, showing that the curved shape of the hill has been determined by the dips. The rocks on Pabini Bie are much darker and less argillaceous.

Exceedingly little concentrated manganese oxide is on any of these hills; they are merely of scientific interest. In this locality are many isolated hills which will be examined when opportunity offers.

ZUARAGU DISTRICT

The deposits in the Zuaragu district are highly siliceous and of no economic value.

VALUE OF THE DEPOSITS

From the foregoing remarks it is apparent that the only deposit of manganese ore of economic importance that is known in the Gold Coast is that of Insuta-Dagwin. It is in a district in the thick forest-region with one of the highest annual rainfalls on the Coast. How far these conditions are necessary for the concentration of the ore is not yet known, but there seems to be no doubt that excessive humidity and protection from loss of soil are two essentials.

Analyses of average ore from some of the foregoing occurrences are as follows:

ANALYSES OF SOME GOLD COAST MANGANESE ORES

	Silica, Per Cent.	Manganese, Per Cent.	Iron, Per Cent.	Phosphorus, Per Cent.
Insuta-Dagwin.....	3 to 7	50 to 53	2 to 4	0.1 to 0.12
Odumase-Konongo.....		22.22	4.37	Trace
Kalimbi Hill.....		15 to 22	5 to 10	Trace
Pongo		30.1	7.7	
Tindongo		9 to 12	2 to 4	Trace

NOTES ON THE PETROLOGY OF CERTAIN ASSOCIATED MANGANESE SILICATE-BEARING ROCKS

BY MAJOR N. R. JUNNER, M. C.

Although manganese-garnets are rarely seen in hand specimens of the manganese ores and associated rocks, they have been detected by the writer in thin sections of all the ores and wall rocks that have been sliced, with the single exception of the Insuta deposit. Rhodonite, and an almost colorless amphibole, which appears to be manganiferous, are less common.

TYPES OF OCCURRENCE

(a) *Rocks Composed of Manganese-garnet and Quartz*

The best known examples of this type are from near Bole and Wa in the Northern Territories of the Gold Coast, and from near Chama in the Gold Coast Colony. Normally these are fine and even-grained rocks, in which the garnets cannot be recognized in hand specimens. They are often stained with secondary manganese oxide. Thin sections examined microscopically are seen to be composed of aggregates of minute crystals of garnet in a matrix of mosaic quartz. Some manganese oxide and braunite (?) is often present, but other minerals are exceptional.

The relative proportions of garnet and quartz in these rocks vary greatly and there is a complete gradation from quartz-rock to garnet-rock.

(b) *Rocks Composed Mainly of Manganese-garnet and Manganese Oxides*

Rocks of this type are known from Kalimbi Hill near Siripe, Pabini Hill near Bangwon, near Odumase and Konongo, and in the Ya Ya creek, about 5 miles east of Dunkwa. These are usually fine-grained banded dark-colored rocks, composed of abundant minute garnets in a matrix of black manganese oxide and braunite (?). A little microcrystalline quartz is often present, and occasionally a little colorless amphibole.

(c) *Rocks Composed of Rhodonite and Manganese-garnet*

Rocks of this type are not common. They are known from near Chama, and from the road between Senyong and Mangkuma. Some quartz is usually present.

All these rocks resemble closely in texture and composition certain rocks from India, described by Fermor¹ as "gondite;" and they also resemble certain rocks described by Derby² from the Queluz district,

¹ L. L. Fermor: The Manganese Ore Deposits of India. *Mem. Geol. Survey India* (1909) 37, Pt. II, 325.

² O. A. Derby: Manganese Deposits of the Queluz District, State of Minas Geraes, Brazil. *Amer. Jnl. Sci.* (1901), 18.

Brazil. The rocks of type (a) correspond to the typical gondite of Fermor; those of type (b) to "spessartite-braunite-rock," and those of type (c) to "rhodonite-spessartite-rock" and "rhodonite-gondite."

PETROLOGY

(a) *Rocks Composed Mainly of Manganese-garnet and Quartz*

Two good examples of this type occur near Bole, one on the road from Bongfu to Wandara, and the other between Tangapi and Bole.

(1) BONGFU TO WANDARA, AT $3\frac{1}{4}$ MILES

At this place a narrow belt of manganese-garnet-quartz-rock, with some psilomelane is associated with quartz-schists containing kyanite and a little sillimanite. These rocks are intruded by fine-grained biotite-granite which outcrops close by.

Specimen No. 11455.—A thin section of this rock examined under the microscope is composed of aggregates of colorless crystals of garnet of uniform size, in a mosaic of quartz showing strain polarization (Fig. 7). Many of the garnets are intersected by more or less parallel veinlets of quartz, which in places appear to be in optical continuity with the quartz grains of the mosaic. Garnet is well in excess of quartz. A little manganese oxide, some of it black and opaque and some brown and translucent, is also present, also very scanty greenish-yellow to colorless fibrous chlorite (?). The garnet is rather cloudy, due to abundant dusty black inclusions. Most of it is unaltered, but slight alteration to brown manganese oxide is visible in parts of the section.

An analysis of this rock and also of the separated garnet are given in Tables 1, 2 and 3.

(2) TANGAPI TO BOLE, AT 1.15 MILES

A narrow band of manganese-garnet-quartz-rock, with some manganese ore occurs in quartz-schist which is underlain by quartz-kyanite-schist, muscovite-quartz-schist and quartzite. Small intrusions of foliated fine-grained biotite-granite, with veins of pegmatite, outcrop within 300 yd. of the manganiferous rocks.

Specimen No. 11529.—This rock is fine and even-grained and is composed of abundant minute idiomorphic colorless garnets of approximately equal size in a mosaic of quartz with a trace of apatite (Fig. 8). Quartz and garnet are present in about equal amounts. The garnets and quartz show strain effects under crossed nicols. Some of the garnets are altered marginally to brown manganese oxide.

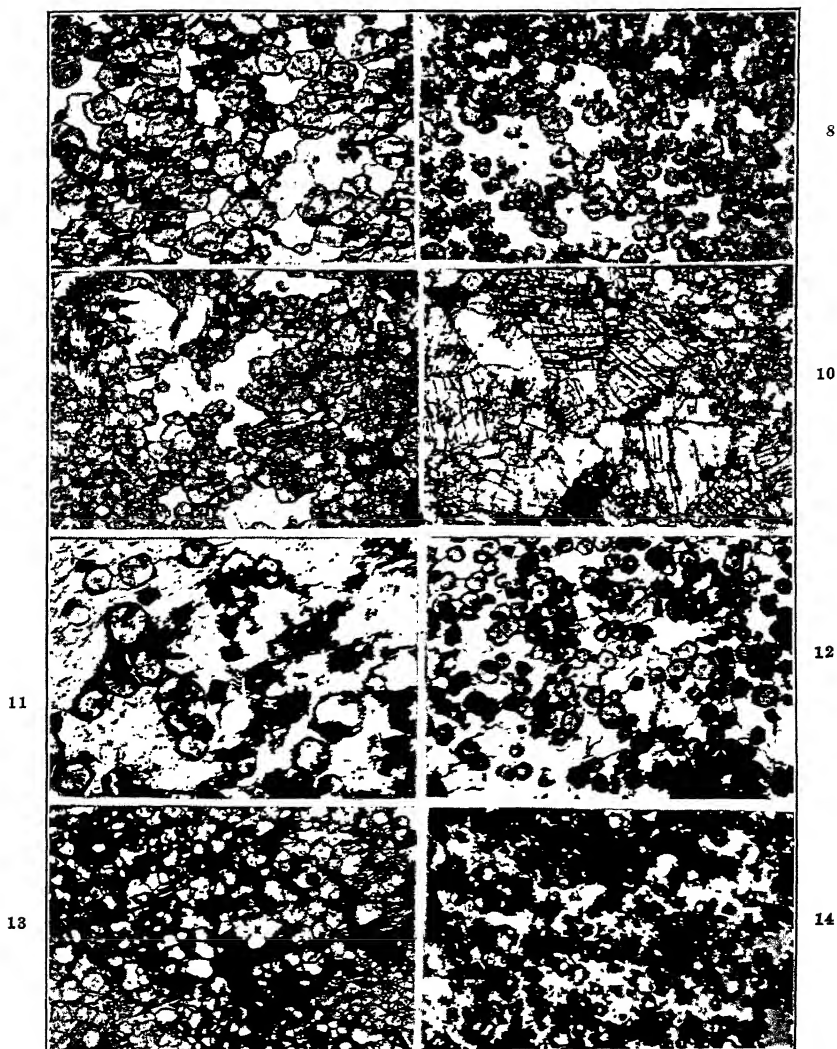


FIG. 7.—SPECIMEN NO. 11455, GRANULAR AGGREGATES OF MANGANESE GARNET AND QUARTZ. BONGFU TO WANDARA AT $3\frac{1}{4}$ MILES. $\times 18$, O. L.

FIG. 8.—SPECIMEN NO. 11529, AGGREGATES OF GARNET IN A BASE OF QUARTZ (WHITE). TANGAPI TO BOLE AT 1.2 MILES. $\times 22$, O. L.

FIG. 9.—SPECIMEN NO. 9239(1), DENSE AGGREGATES OF MINUTE MANGANESE GARNETS WITH QUARTZ (WHITE) AND LITTLE COLORLESS AMPHIBOLE. YABIO. $\times 16$, O. L.

FIG. 10.—SPECIMEN NO. 12093, RHODONITE AND SUBORDINATE MANGANESE GARNET AND A TRACE OF QUARTZ. NUMEROUS SMALL GARNETS SEEN ON RIGHT-HAND SIDE OF THE PHOTOGRAPH. SANYONG TO MANGKUMA AT 5.1 MILES. $\times 18$, O. L.

(3) SIRO TO WA, AT 4.85 MILES

The rocks are manganiferous fine mica-schist and some quartzites, of pale to dark gray and black colors. Some of the rocks are highly charged with manganese oxide, and in places masses of manganese ore are visible on the surface.³

Specimen No. 11919.—This rock is thinly banded and fine grained and resembles a sandstone. With the aid of a magnifying glass abundant very small black manganese garnets can be recognized. The specific gravity of the rock is 2.53.

Examined under the microscope the rock is seen to be composed of a matrix of mosaic quartz, through which is scattered a large number of small idiomorphic colorless garnets in all stages of the change to manganese oxide (Fig. 11). In addition there are present occasional small irregularly shaped patches of manganese ore, which is black and opaque in reflected light and partly brown and translucent in transmitted light. These patches apparently represent original manganese oxide, which has been recrystallized during the metamorphism of the rock. A good deal of finely divided manganese oxide occurs in minute rods and granules through the quartz. The quartz is well in excess of the garnet. Odd very small crystals of zircon were also noted.

Specimen 11921 is a banded dark-colored fine-grained rock, containing abundant very minute garnets which are pink when fresh and black when altered (Fig. 12). The specific gravity is 2.78. Under the microscope the rock is composed of a large number of small idiomorphic colorless garnets in a matrix of mosaic quartz which shows strain effects in polarized light. The quartz is slightly in excess of the garnet. The garnets, although closely packed, are for the most part isolated. Black dusty inclusions are common in the garnet. All stages in the break-down of the garnet are visible. The alteration which starts from the edges of the garnet crystals is usually to brown and black manganese oxide, but some of the garnets have been replaced by a colorless isotropic mineral with a low refractive index. Occasional irregularly shaped patches of black opaque manganese oxide, which do not appear to have been formed from the break-down of the garnet, are also present.

³ Annual Report on the Gold Coast Geological Survey (April, 1923 to March, 1924) 23.

FIG. 11.—SPECIMEN NO. 11919, MANGANESE GARNET IN A BASE OF QUARTZ. THE ALTERATION OF THE GARNET TO MANGANESE OXIDE IS WELL SEEN. SIRO TO WA AT 4.85 MILES $\times 16$, O. L.

FIG. 12.—SPECIMEN NO. 11921, MANGANESE GARNET, QUARTZ AND MANGANESE OXIDE (BLACK). SIRO TO WA AT 4.85 MILES $\times 18$, O. L.

FIG. 13.—SPECIMEN NO. 9239 (2), CLOSELY PACKED GARNET AGGREGATES BREAKING DOWN TO MANGANESE OXIDE (BLACK). YABIO. $\times 16$, O. L.

FIG. 14.—SPECIMEN NO. 11487, MANGANESE OXIDE (BLACK) DERIVED FROM INNUMERABLE MINUTE GARNETS. THE WHITE BACKGROUND IS ESSENTIALLY QUARTZ. KALIMBI HILL. $\times 32$, O. L.

(4) YABIO, NEAR CHAMA

In this district rocks containing manganese-garnet, and in some cases rhodonite, are associated with coarsely crystalline metamorphic rocks, such as amphibolites, hornblende-pyroxene-gneiss and biotite-schists and gneisses.

Specimen 9239 is a heavy aphanitic textured, pale pink rock, with a little black manganese oxide. A thin section of the rock is composed of compact aggregates of very minute colorless garnet, with a little interstitial quartz and colorless amphibole. At one side of the section is a veinlet of quartz containing some amphibole and a fairly large crystal of a colorless uniaxial mineral, the optical properties of which agree with apatite. Along the contact between the veinlets of quartz and the garnet aggregates a good deal of colorless amphibole is seen (Fig. 9). The breaking down of the garnet to black and brown manganese oxide is well seen in parts of the section (Fig. 13).

(b) *Rocks Composed Mainly of Manganese-garnet and Manganese-oxide*
(1) ODUMASE AREA

The manganiferous rocks are here associated with phyllites, greenstones, chloritic argillaceous schist and a minor amount of sandy strata.⁴

Specimen No. 13825. Juansa to Konongo; South of Birrinsim Hill at $\frac{3}{4}$ Mile East of Motor Road.—Examined under the microscope this is a banded rock, consisting of innumerable small idiomorphic colorless garnets of even size, in a matrix of finely divided manganese oxide and subordinate cryptocrystalline to microcrystalline quartz, with lines of black opaque dust, which is manganese oxide or possibly carbon. The manganese oxide is brownish black and even brown on thin edges in transmitted light, and has a light gray metallic lustre in reflected light. The garnet contains some minute inclusions and shows some alteration around the edges to brownish black manganese oxide. The bulk of the manganese oxide in this rock has not been derived from the alteration of the garnet. It represents original manganese oxide in the rock before metamorphism.

Specimen No. 13780. Odumase-Adwiade Track, at $\frac{3}{4}$ Mile Southwest of the Railway Section.—A thin section of this rock consists of bands of brownish black manganese oxide, crowded with isolated minute colorless, euhedral garnets, and an occasional lighter colored band containing abundant minute garnets, with some finely divided quartz and dusty and finely granular manganese oxide. As in No. 13825, the bulk of the manganese oxide has not been derived from the alteration of the manganese-garnet.

⁴ Reports on Geological Survey of the Gold Coast. Jan., 1922, to March, 1923, p. 37, and April, 1924, to March, 1925, p. 24.

(2) HAPA LOCALITY

The rocks associated with the manganese deposits are siliceous and argillaceous slates.⁵

Specimen No. 12449. Pabini Bie Hill, South Southwest of Hapa.—This is a heavy, black, very fine-grained, granular rock. A thin section is composed of innumerable minute almost colorless garnets, cemented together by black opaque manganese oxide. Small patches consisting of aggregates of minute needles of colorless amphibole in a base of quartz grains are also present. Most of the amphibole needles are stained by brown manganese (?) oxide. The amphibole is possibly manganiferous. Veinlets of quartz traverse the section.

(3) KALIMBI HILL, NEAR SIRIPE

The hill consists of manganiferous sediments which have been intensely disturbed, contorted and fractured. Veins of quartz traverse the sediments, and on the western side of the deposit a narrow outcrop of biotite-granite extends nearly to the foot of the hill. On the northeast side, near the foot of the hill, actinolite-bearing rocks and granodiorite outcrop.⁶

Specimen No. 11487 from Kalimbi Hill is a contorted very fine-grained, thinly banded, brownish black, porous rock, with veinlets of quartz (Fig. 14). The specific gravity is 2.63. Examined microscopically the rock is seen to be composed essentially of brownish black manganese oxide, with a little quartz and a trace of rutile and feldspar. The manganese oxide is derived almost wholly from innumerable very minute idiomorphic garnets. A small vein of quartz traverses the section.

(c) *Rocks Containing Rhodonite in Addition to Manganese-garnet*

Specimen No. 12093. Senyong to Mangkuma, at 5.1 Miles.—A thin section of this rock is composed essentially of closely packed allotriomorphic phenocrysts of rhodonite, with subordinate very small idiomorphic manganese-garnets and a little colorless amphibole, sphene, quartz and a trace of carbonate (Fig. 10). The rhodonite contains inclusions of garnet and quartz. The rhodonite is colorless and its refractive index is well below that of the garnet. The polarization colors are low-order grays and yellows. The rock is fairly fresh, but in areas of manganese-garnet some alteration to brown manganese oxide is visible. This rock is identical with the "rhodonite-spessartite-rock" of Fermor.⁷ An analysis of it is given in Table 4.

⁵ Annual Report of the Geological Survey of Gold Coast (April, 1923, to March, 1924) 28.

⁶ Annual Report on the Geological Survey of the Gold Coast (April, 1923, to March, 1924) 49.

⁷ L. L. Fermor: *Op. cit.*, 326.

CHEMICAL COMPOSITION

(a) *Manganese-garnet-quartz-rock*

The presence of manganese dioxide in this rock from Bongfu to Wandara at $3\frac{1}{4}$ miles (No. 11455, Table 1⁸) cannot be determined chemically because of the presence of excess ferrous oxide. If manganese dioxide were present, the amount of ferrous oxide would be correspondingly increased and the ferric oxide decreased.

Typical gondite from Wagora, Chindwára district, India, is composed of about equal volumes of garnet and quartz.⁹

The lower percentage of silica and the higher percentage of manganous oxide in the Gold Coast specimen are due to the garnet being in excess of the quartz, as shown in the calculated mineral composition given below. Apart from the percentages of SiO₂ and MnO the analyses of the rocks are similar.

The mineral compositions calculated from the analyses are given in Table 2.

TABLE 1.—*Chemical Analyses of Specimens of Manganese-garnet-quartz-rock in Percentages*

	SPECIMEN No. 11455	GONDITE FROM WAGORA, INDIA ¹⁰
SiO ₂	49.79	60.45
Al ₂ O ₃	14.69	12.36
Fe ₂ O ₃	5.16	7.47
FeO.....	2.09	0.13
MgO.....	2.04	1.90
CaO.....	4.69	2.60
Na ₂ O.....	0.18	0.19
K ₂ O.....	0.11	0.03
H ₂ O+.....	0.85	0.20
H ₂ O—.....	0.23	0.10
CO ₂	Nil	
TiO ₂	0.19	0.46
P ₂ O ₅	0.16	0.68
Cl.....	Nil	Trace
S.....	0.02	
Cr ₂ O ₃	Nil	
V ₂ O ₅	0.05	
NiO.....	Nil	
MnO.....	20.26	13.37
BaO.....	Trace	
CuO.....		0.02
Totals.....	100.51	

⁸ H. L. Riley: *Analyst*.

⁹ L. L. Fermor: *Op. cit.*, 349.

¹⁰ L. L. Fermor: *Op. cit.*, 350.

TABLE 2.—*Mineral Composition of Specimens of Manganese-garnet-quartz-rock, in Percentages*

Mineral	Specimen 11455	¹¹ Gondite from Wagora, India
Quartz.....	23.1	39.85
Spessartite.....	47.0	
Grossularite.....	10.0	
Andradite.....	3.0	54.89
Almandine.....	4.9	
Pyrope.....	6.8	
Limonite.....	5.1	
Na ₂ O, K ₂ O, excess H ₂ O.....	0.9	
Total.....	100.8	

The chemical composition of garnet from several localities is given in Table 3 with comments on each specimen following the table.

TABLE 3.—*Chemical Composition of Different Specimens of Garnet, in Percentages*

	In Specimen No. 11455, as Analyzed (1)	In Specimen No. 11455, Calculated (2)	In Gondite from Wagora, India (3)	Queluz, Brazil, Garnet (4)	Broken Hill	
					Pink Garnet (5)	Brown Garnet (6)
SiO ₂	39.75	37.1	37.73	38.47	36.32	36.80
Al ₂ O ₃	19.94	21.1	21.26	21.07	20.02	20.48
Fe ₂ O ₃	3.76	4.0	.	7.38	1.60	2.37
FeO.....	4.54	4.8	9.94		12.93	19.23
MgO.....	1.51	1.6	3.48		0.23	0.06
CaO.....	5.53	5.8	3.11	4.70	3.10	2.80
MnO.....	24.25	25.6	24.48	27.90	25.16	17.82
NiO.....	Nil					
H ₂ O+.....	0.54					
H ₂ O-.....	0.01					
TiO ₂	0.26				0.47	0.23
	100.09	100.0	100.00	99.52	99.83	99.79

(b) *Chemical Composition of Garnet*

The garnet in specimen No. 11455 from Bongfu to Wandara, at 3¼ miles (Column 1), was separated from the crushed rock by heavy liquids, by the use of an electromagnet and by careful handpicking under the microscope. The garnet so separated was freed from manganese stain by gentle treatment with HCl.¹²

¹¹ L. L. Fermor: *Op. cit.*, 350¹² H. L. Riley: *Analyst*.

Column 2 gives the percentage composition of the garnet in No. 11455, obtained by calculating the mineral composition from the analysis (Column 1) and recalculating the garnet portion to 100 per cent. In analysis (1) there is a deficiency of RO to satisfy R_2O_3 in garnet, and the Fe_2O_3 is therefore used as FeO.

Column 3 shows the calculated percentage composition of the garnet in typical gondite from Wagora, Chindwára district, India.¹³

Column 4 is the analysis of a rock composed almost exclusively of garnet from the Piquiry deposit of manganese ore—Queluz district—State of Minas Geraes, Brazil.¹⁴

Column 5 gives the analysis of pink garnet from "garnet sandstone," in 500-ft. level, Central Mine, Broken Hill.¹⁵

Column 6 is the analysis of brown garnet from "garnet sandstone" in 500-ft. level, Central Mine, Broken Hill.¹⁶

From the analyses it will be seen that the garnets from the Gold Coast, India and Brazil are very similar in composition. These garnets are not pure spessartite. In the Gold Coast example the garnet is composed by weight of about 60 per cent. spessartite, 19 per cent. almandine, 15 per cent. grossularite and 6 per cent. pyrope. The garnets from Broken Hill are richer in iron and the brown garnet belongs to the almandine-spessartite type.

(c) *Chemical Composition of Rhodonite-spessartite-rock.*

Table 4 gives the result of a complete analysis¹⁷ of specimen No. 12093, from between Senyong and Mangkuma, the calculated mineral composition of the rock and also of the rhodonite.

Because of the presence of excess manganese dioxide in No. 12093 it is impossible, chemically, to estimate the amount of ferrous oxide present, Column (1). The figure given for the manganese dioxide represents the excess of manganese dioxide over the ferrous oxide present.

For the calculation of this mineral composition, Column (2), we assume that the garnet in 12093 has the same composition as the garnet in 11455, and that the Fe_2O_3 is present as FeO.

The percentage composition of the rhodonite, Column 3, is calculated from the composition of the rhodonite obtained in Column (2).

¹³ L. L. Fermor: *Op. cit.*, 351.

¹⁴ O. A. Derby: *Op. cit.*, 18.

¹⁵ E. C. Andrews: The Geology of the Broken Hill District, *Mem. Geol. Survey, N. S. Wales*, Geology No. 8. (1922). Appendix II, by F. L. Stillwell, 380.

¹⁶ F. L. Stillwell: *Op. cit.*, 380. (See reference ¹⁴.)

¹⁷ H. L. Riley: *Analyst*.

TABLE 4.—*Chemical Analysis and Mineral Composition of Specimen No. 12093, in Percentages*

(1) Percentage Composition		(2) Calculated Mineral Composition			(3) Calculated Composition of the Rhodonite	
SiO ₂	46.18	Quartz.....	7.0	7.0	SiO ₂	48.4
Al ₂ O ₃	3.25				FeO.....	2.9
Fe ₂ O ₃	3.79	Spessartite.....	9.5	15.5	CaO.....	7.1
FeO.....	Nil	Grossularite....	2.4		MgO.....	5.1
MgO.....	3.81	Pyrope.....	1.0		MnO.....	36.5
CaO.....	6.77	Almandine.....	2.6	69.5		100.0
Na ₂ O.....	Trace					
K ₂ O.....	0.05	CaSiO ₃	10.2			
H ₂ O+.....	1.12	MgSiO ₃	8.9			
H ₂ O-.....	0.38	FeSiO ₃	3.7			
CO ₂	0.57	MnSiO ₃	46.7			
TiO ₂	0.35					
P ₂ O ₅	0.23	MnO ₂		4.0		
Cl.....	Trace	Calcite.....		1.3		
S.....	0.12	Apatite.....		0.4		
NiO.....	Nil	Rutile.....		0.4		
MnO.....	29.26	H ₂ O, S, etc....		1.7		
				99.8		
MnO ₂	3.99					
BaO.....	Nil					
	99.87					
Correction for S.....	-0.04					
	99.83					

ORIGIN

The field relations and the microscopical characters of these rocks show that they are metamorphosed sediments. Before metamorphism the sediments consisted of argillaceous matter, free silica and manganese oxide in varying proportions, with small amounts of lime, and oxides of iron, magnesium, titanium, phosphorus, sodium and potassium. The sediments were unusually low in alkalis.

The composition of the original sediments determined the nature of the final rock. Where the sediment was a manganiferous clay, spessartite was readily formed. If all the MnO and Al₂O₃ were used up in the process, any excess of silica would recrystallize as quartz. If after the formation of spessartite there were an excess of MnO and SiO₂, manganese oxide and quartz or possibly some braunite and rhodonite would form. Where Al₂O₃ was in excess of the MnO necessary for garnet, an aluminum silicate, such as kyanite, was formed.

The explanation here advanced for the origin of these rocks is the same as that proposed by Fermor¹⁸ for the Indian "gondites," which so closely resemble the Gold Coast rocks. The somewhat similar rocks of the Queluz district of Brazil, which consist for the most part of mixtures of manganese-garnet, quartz and manganese oxides, are regarded by O. A. Derby¹⁹ as having been derived from amphibolic schists representing sheared basic eruptives, but the evidence for this conclusion does not seem very convincing.

Enough detailed work has not been done on the Gold Coast rocks to define the metamorphic conditions necessary for the generation of the spessartite and rhodonite. The spessartite develops readily in places where the metamorphism is of a low grade and also occurs where the metamorphism is more intense. In this connection Tilley²⁰ states that in pelitic stress-zones "the presence of significant percentages of MnO in the sediment has a profound influence on the order of entry of the index minerals in progressive metamorphism, the common order chlorite-biotite-garnet being inverted to chlorite-garnet-biotite." Rhodonite, on the other hand, has been found only in areas of intense metamorphism, although in other areas the composition of the manganese rocks was apparently favorable for the formation of rhodonite.

In conclusion I wish to express my thanks to F. Oates for checking the calculations of the mineral compositions of the rocks and of the garnet.

DISCUSSION

J. W. FURNESS, Washington, D. C.—The Gold Coast has taken the position of the third largest producer of manganese in the world. Its geographical position makes it evident that neither India nor the Caucasus, without considering Brazil, can eliminate the Gold Coast ores from the world market, even by a reduction in price.

The present commercial importance of the Insuta-Dagwin manganese deposits may be better appreciated when one considers the world's manganese situation prior to their discovery. In 1898 the world's production was 200,000 tons, chiefly derived from Russia, India and Brazil, and in 1926, the estimated world's production was 2,500,000 tons. With the advent of the open-hearth furnace the consumption of manganese was greatly increased and the known occurrences in the countries mentioned were able to meet the demand with comparative ease. For many years keen competition existed between the manganese producers of India and those of the Caucasus. With the increased production of Brazil this competition was rendered more acute. For almost a decade prior to 1914 the world's price of manganese, in spite of increased demand, remained relatively constant.

The reserves of the three sources mentioned, it may be well to state, are of such magnitude that they will unquestionably meet the world's demand for many years.

¹⁸ L. L. Fermor: *Op. cit.*, 290.

¹⁹ O. A. Derby: *Op. cit.*, 30.

²⁰ C. E. Tilley: On Garnet in Pelitic Contact Zones. *Mineralog. Mag.* (London, June, 1926) 48.

During the war period the inadequacy of the domestic mines to supply the needs of the United States was accentuated. The cessation of exports from Russia threw the burden of the world's demand upon India and Brazil. Intensive search for new deposits was world-wide in its scope.

In 1914, Mr. Kitson, Government Geologist of the Gold Coast, discovered the Insuta-Dagwin deposits. The reserves of detrital ore have been estimated by the company²¹ to be 3,000,000 tons. The ore *in situ* has been estimated to be in excess of 10,000,000 tons. It seems probable from what we have learned that these estimates are most conservative. The predominating mineral of the ore is psilomelane with occasional pieces of pyrolusite. The grade of the ore is comparable with that of the other great sources, the iron content being less than 5 per cent., silica less than 8 per cent., and phosphorus below 0.15 per cent. In 1916 the deposits shipped were 4228 tons. In 1926, the estimated shipments were slightly in excess of 360,000 tons, or approximately 14 per cent. of the world's consumption. The deposits are 33 miles from tidewater, and are connected with the harbor of Sekondi by rail.

In 1922 a British corporation known as the African Manganese Mining Co., Ltd., was formed to exploit this deposit. In late years the larger part of the production has been shipped to Norway and Canada, where it is manufactured into ferromanganese by the Electro-Metallurgical Sales Corp'n. (American). A considerable quantity of crude ore finds its way annually into the world's market as high-grade manganese ores. Upwards of 85 per cent. of the ferromanganese imported by the United States in 1926 came from these sources.

In July, 1925, the Harriman interests acquired a lease on the Tchiatouri deposit of Georgia. The Bethlehem Steel Corp'n. began to purchase these ores and in 1926 placed ferromanganese on the market. Prior to that time, the company manufactured ferromanganese for its own consumption. The result of this action was what might be termed a "trade war," and the price of ferromanganese, duty paid Atlantic seaboard, dropped from \$115 a ton to \$80 a ton. It has since advanced to \$100 a ton. The importance of the Gold Coast is indicated by this illustration.

²¹ The Fanti Consolidated Co., see bottom p. 379 and top p. 380. As regards the ore *in situ* I know no such estimate. My own estimates of both kinds of ore for our Government cannot be given as they are confidential.—Sir Albert E. Kitson.

The Importance of Manganese in the Steel Industry

By H. M. BOYLSTON,* CLEVELAND, OHIO

(Cleveland Meeting, April, 1927)

METALLIC manganese was first produced in 1773, by Sven Rinmann, a Swedish mineralogist. In 1799, William Reynolds, of Ketley, England, obtained a patent on the use of manganese dioxide in the manufacture of steel, and in 1808 an English patent was granted to John Wilkinson covering the use of manganese ore, or manganiferous ore, as an addition to iron ore in the blast furnace for the production of a superior grade of pig iron. Karsten showed in 1816 that manganese increased the hardness of iron without decreasing its malleability or toughness, while it caused pig iron to be more white in fracture and more fusible.

The first practical application of manganese to steel-making, however, is attributed to Josiah Heath who discovered in 1839 (less than 90 years ago) that manganese conferred upon steel the properties of being more easily worked and welded under the hammer. When Bessemer invented the pneumatic process of steel-making which now bears his name he was at first successful because, as it afterward developed, he was using a Swedish pig iron high in manganese. When his supply of this material ran out, he used a low-manganese pig from English sources and failed signally to produce a steel which could be rolled, while his Swedish licensees were continuing their successful application of his method. And it was not until Robert Mushet, in 1856, suggested the addition of manganese in the form of spiegeleisen to the molten product of the converter that success again came to Bessemer. In 1866, William Siemens patented the use of ferromanganese as an addition to molten steel to counteract the bad effects of sulfur.

It was in 1888 that Robert Hadfield published at some length his account of the effect of manganese as an alloying element in steel and described the unique properties of the now famous Hadfield manganese steel containing about 12 to 14 per cent. manganese and 1.25 per cent. carbon.

FUNCTION OF MANGANESE IN STEEL-MAKING

Bessemer's failure at one period of his experimental work to make steel which could be forged successfully was due to the fact that the

* Professor of Metallurgy, Case School of Applied Science.

severely oxidizing conditions under which his process was conducted caused the existence in his finished metal of some oxide of iron (especially when his pig iron was low in manganese). This required the later addition of some material like manganese, which was the material suggested by Mushet, to deoxidize the metal, because iron oxide, even in quite small quantities, makes steel both red short and cold short. Ledebur has stated that iron containing combined oxygen up to 0.10 per cent. can be worked, but that above this the metal is bad. The addition of manganese counteracts this trouble and later on it was discovered that it counteracts also the red shortness due to the presence of small amounts of sulfur. In the ladle and more especially in the mixer the sulfur content of the pig iron prepared for steel-making may actually be reduced by the formation of manganese sulfide (MnS), which then floats to the surface in part and mixes with the slag.

Now sulfur, like oxygen, makes steel red short, and as little as 0.02 per cent., in the form of iron sulfide (FeS), will produce that effect, at least within certain temperature ranges, and even in the presence of as much as 0.03 per cent. manganese. I have in mind here the commercially pure form of iron known as ingot iron which with those quantities of sulfur and manganese must be allowed to rest undisturbed by rolling during a range of some 150°C ., where the tenderness occurs, after which it can again be rolled to the final size. Theoretically, it requires only about 1.7 times as much manganese as sulfur present in steel to convert the FeS into MnS , which is relatively innocuous compared with FeS as regards red shortness, but actually it requires about 8 times as much manganese as sulfur to accomplish this result, so that with the usual upper limit of 0.05 per cent. sulfur allowed in ordinary structural steel, it is customary to add enough manganese during recarburization to deoxidize the steel and to leave from 0.40 to 0.50 per cent. of residual manganese in the steel to offset the bad effect of sulfur.

MANGANESE AS AN ALLOYING ELEMENT

In addition to its valuable deoxidizing and desulfurizing qualities and the fact that it renders the remaining sulfur comparatively innocuous, manganese also confers upon steel valuable physical properties of its own. It increases the elastic ratio and the ultimate strength without appreciably lowering the ductility, in much the same manner as nickel, and apparently requires only about half as much manganese as nickel to produce the same effect so that the expense of certain structural steels can be appreciably reduced by substituting manganese for nickel. Moreover, the increase in physical properties due to heat treatment is more noticeable when considerable quantities of manganese are used.

TABLE 1.—*Tests Illustrating Effect of Manganese on Steel*

	Effect as an Alloying Element on Tensile Strength of Structural Steel			Effect on Yield Point of Steel			Effect on Percentage Elongation in 2 In. and Reduction of Area			Effect on Resistance to Impact (Charpy Test)		
	A	B	C	A	B	C	A	B	C	A	B	C
	T. S., Lb. per Sq. In.			Yield Point, Lb. per Sq. In.						Ft. Lb. Necessary to Fracture		
	A	B	C	A	B	C	A	B	C	A	B	C
Condition of steel, percentage of manganese....	0.51	1.43	1.78									
As received.....	75,500	105,750	119,000	49,500						22.96	47.6	14.4
Air cooled from 1550° F.....		80,090	100,025	54,400							63.0	27.6
Furnace cooled from 1537° F.....	70,000			41,000						15.20		
Water quenched from 1425° F. ¹	135,500	197,600	234,925	165,500	165,000		30.0:51.9	34.4:70.35	26.6:54.70	10.94	10.6	6.0
Water ² quenched from 1425° F. ¹ Drawn at 600° F. ³	100,500	178,200	199,250	68,000	174,200	180,250	8.0:16.9	2.8:	2.35	10.24	16.0	5.0
Water ³ quenched from 1425° F. ¹ Drawn at 925° F. ⁴	98,000	120,300	134,150	67,000	111,450	125,300	2.3:	2.58	4.1	19.86	49.0	46.4
Water ² quenched from 1425° F. ¹ Drawn at 1200° F. ⁵	84,000	94,125	108,450	61,500	78,025	97,550	30.0:71.4	25.3:63.56	24.3:64.66	26.30	67.0	63.0

¹ Steel A was quenched at 1537° F.² Steel A was quenched in oil and not in water.³ Steel A was drawn at 707° F.⁴ Steel A was drawn at 860° F.⁵ Steel A was drawn at 1200° F.

The results shown for steel A were obtained from the paper, "Effect of Carbon on the Physical Properties of Heat-treated Carbon Steel," by J. H. Nead, *Trans.* (1916) 53, 218. The results for steels B and C were obtained in the author's laboratory, Case School of Applied Science, in 1926, by K. R. Van Horn and E. C. Knuth. The data for the Charpy impact test results on Steel A were from an article by F. C. Laugenberg, A. S. T. M. (1921) 21, 849.

Several steel companies, notably the Bethlehem Steel Corp., have for some years now been making a steel containing from 0.30 to 0.50 per cent. carbon and 1.25 to 1.50 per cent. manganese, while more recently there has appeared on the market a steel containing some 0.25 per cent. carbon and 1.40 to 1.80 per cent. manganese, and it is understood that some of the automobile manufacturers are using the latter for such critical parts as steering spindles and rear axle shafts.

In Table 1 will be found some typical physical properties of rolled steels containing respectively from 0.25 to 0.30 per cent. carbon and 0.51 to 1.78 per cent. manganese.

COMPOSITION OF STEEL IN TABLE 1

	CARBON, PER CENT.	MANGANESE, PER CENT.	SILICON, PER CENT.	PHOSPHORUS, PER CENT.	SULFUR, PER CENT.
A.....	0.32	0.51	0.128	0.009	0.027
B.....	0.27	1.43		0.030	0.029
C.....	0.25	1.78		0.042	0.025

During the last 10 years many steel castings have been made containing 0.25 to 0.35 per cent. carbon and about 1.50 per cent. manganese for use as anchor chains, car coupler knuckles, etc. In Table 2 are given typical physical properties of such castings in comparison with steel castings of plain carbon steel of about the same carbon content.

TABLE 2.—*Physical Properties of Steel Castings Showing Influence of Manganese*

Percentage composition of steel:

Carbon.....	0.27	0.35
Manganese.....	0.60	1.50
Tensile strength, lb. per sq. in.....	67,000 to 75,000	90,000 to 130,000
Elastic limit.....	25,000 to 53,000	About 60 per cent. of T. S.
Percentage elongation in 2 in.....	24.5 to 38	10 to 25
Percentage reduction of area.....	29 to 61.7	20 to 50

The variation in properties is due to variations in heat treatment.

Some typical figures for the physical properties of water-toughened manganese steel castings are shown in Table 3.

TABLE 3.—*Physical Properties of Steel Castings Containing about 12 Per Cent. Manganese and 1.25 Per Cent. Carbon (Hall)*

Tensile strength, lb. per sq. in.....	80,000 to 110,000
Elastic limit, lb. per sq. in.....	35,000 to 45,000
Percentage elongation in 2 in.....	15 to 35
Percentage reduction of area.....	15 to 35
Degrees bend, $\frac{1}{2}$ by 1 in.....	180
Brinell hardness.....	180 to 200

ECONOMIC IMPORTANCE OF MANGANESE IN STEEL MAKING

Assuming for the moment that all the steel manufactured contains at least 0.40 per cent. of manganese on the average (and this ignores completely the use of manganese as an alloying element) and allowing a loss of 50 per cent. of manganese added to the steel for deoxidizing and desulfurizing purposes, there will be needed for each ton of steel produced something like 16 lb. $\left(\frac{0.40 \times 100 \times 2000}{50 \times 100} = 16 \right)$ of manganese

in the form of ferromanganese. The latter contains on an average about 80 per cent. metallic manganese so that there will be needed for steel-making about 20 lb. of 80 per cent. ferromanganese per net ton, or 22.40 lb. per gross ton, or 1 per cent. of the total weight of steel produced. In 1925 there were 45,393,524 gross tons of steel ingots and castings produced in the United States. Thus it follows that 453,935 gross tons of 80 per cent. ferromanganese was needed in that year alone.

The Annual Statistical Report of the American Iron and Steel Institute for the year 1925 shows that the consumption of ferromanganese in that year was as follows:

	GROSS TONS
Ferromanganese produced.....	260,029
Add imports for consumption.....	77,731
	337,760
Exported.....	5,496
Consumption.....	332,264

In addition there was 66,845 gross tons of spiegeleisen consumed. This alloy contains about 20 per cent. of manganese.

If we assume an average manganese content in spiegel of 20 per cent., 66,845 gross tons of spiegel would contain 13,369 gross tons of metallic manganese. If this were reckoned in terms of 80 per cent. ferromanganese the spiegel could be reckoned as 16,711 gross tons of ferromanganese and the consumption for 1925 in terms of ferromanganese or its equivalent would be:

	GROSS TONS
Ferromanganese consumed.....	332,264
Spiegel converted to terms of 80 per cent. ferromanganese....	16,711
Total ferromanganese or its equivalent.....	348,975

As compared with 453,935 gross tons of ferromanganese called for by the author's calculations there is a discrepancy of 104,960 gross tons. Some of this amount may undoubtedly be accounted for by residual manganese in the open-hearth bath before deoxidation and

recarburization, and some of it is probably due to the growing practice of using a pig iron high in manganese (about 2 per cent.) by means of which a high residual manganese (say 0.25 to 0.30 per cent.) may be kept in the open-hearth bath, thus cutting down quite appreciably the necessary additions of manganese at time of recarburization. Then too, some substitutes for ferromanganese have been used to some extent, such as silico-manganese containing 50 per cent. manganese and 25 per cent. silicon, another grade of mixed alloy known as silico-spiegel, containing 30 to 35 per cent. manganese and 15 to 20 per cent. silicon, and an alloy known as S. A. M. metal which contains about 10 per cent. each of manganese, silicon and aluminum.

MANGANESE. SITUATION IN THE UNITED STATES IN 1926

In conclusion the writer would call attention to an article appearing in the *Skillsings' Mining Review* for Jan. 29, 1927, entitled "Manganese Situation in U. S. in 1926." After presenting figures as to the amount and value of manganese ore constituting domestic shipments during that year, and calling attention to increased imports of manganese ore, including imports from Cuba, Russia, Brazil, British India and the Gold Coast of West Africa, it remarks that "the apparent outstanding feature of the year has been the increase in the domestic manufacture of ferromanganese. During the year the Bethlehem Steel & Iron Co. placed ferromanganese on the market. This action resulted in a decrease in price from \$115 to \$88. At present ferromanganese is quoted at \$100 c. i. f. Atlantic seaboard, duty paid. With the exception of the ferromanganese of Swedish manufacture the present price at which ferromanganese is selling is said to be below the cost of manufacture in foreign countries."

ACKNOWLEDGMENTS

The results for the steel high in manganese were supplied by H. A. Schwartz, director of research, National Malleable & Steel Castings Co., and those for the plain carbon steel were obtained by Prof. Anthony Jenkin and H. C. Lash, in the author's laboratory at Case School of Applied Science. The figures in Table 3 were furnished by J. H. Hall, Taylor-Wharton Iron & Steel Co.

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 H. D. Hibbard: Manufacture and Uses of Alloy Steels. John Wiley & Sons. Inc., New York, N. Y. (Chapter on "Manganese Steels," p. 24; bibliography, p. 39.)
 See also *Bur. Mines Bull.* No. 100 (1915). (Chapter on "Manganese Steels," p. 22).

DISCUSSION

J. STRAUSS, Washington, D. C. (written discussion).—Professor Boylston has indicated briefly the improvement in physical properties due to increase in the proportion of manganese over that present in the usual "carbon steels." The Naval Gun Factory has made considerable use of steels of this class during the past few years in the form of bar stock up to 5 in. diameter. Some typical results obtained on 1-in. round bars are quoted below and check Professor Boylston's data quite well.

TABLE 4.—*Typical Results on One-inch Round Bars*

Quenched and tempered at 900° F.			
	C—0.51 Mn—1.35	C—0.34 Mn—1.88	C—0.43 Mn—2.01
Prop. limit, lb. per sq. in.....	140,000	122,000	147,000
Tensile strength, lb. per sq. in.....	167,000	143,000	166,000
Elongation, per cent. in 2 in.....	13	16	13
Reduction area, per cent.....	42	55	43
Quenched and tempered at 1200° F.			
Prop. limit, lb. per sq. in.....	92,000	87,000	82,000
Tensile strength, lb. per sq. in.....	112,000	105,000	109,000
Elongation, per cent. in 2 in.....	22	23	22
Reduction area, per cent.....	61	67	61

* This same influence of manganese is brought to bear in other alloy steels but in some cases it is noteworthy that the manganese has been omitted from the common name applied to these steels. The ordinary carbon steels in general contain not over 0.70 per cent. manganese and rarely over 0.80 per cent. Steels containing more than this amount should have the presence of the manganese noted in the term by which they are designated, for in such cases manganese is influential in imparting strength and other properties to the combination. This applies in such cases as the so-called "carbon-vanadium" steels which are actually manganese-vanadium steels; also to many of the silicon structural steels about which there has been so much recent agitation in Germany and which are actually silico-manganese steels (1 per cent. Si, 1 per cent. Mn). On the other hand, the spring and gear steels designated by the term "silico-manganese steels" actually contain not more than the normal amount of manganese that is common in simple carbon steels, and therefore would be more truthfully described as silicon steels.

H. M. BOYLSTON (author's reply to discussion).—It is interesting to learn that Mr. Strauss has had some experience with steels containing between 1.35 and 2.01 per cent. manganese. It is quite evident also that others are beginning to realize the important applications of alloy steels containing much lower percentages of manganese than is found in Hadfield manganese steel and which until recently were very difficult to manufacture because of the scarcity of low-carbon ferromanganese. I quite agree with Mr. Strauss in all the points he brings up.

Some Characteristics of Low-carbon Manganese Steel*

By V. N. KRIVOBOK,† B. M. LARSEN,‡ W. B. SKINKLE§ AND W. C. MASTERS,||

PITTSBURGH, PA.

(New York Meeting, February, 1927)

THE study and use of low-carbon manganese steels have been curiously neglected in the general history of developments in alloy steels. Hadfield¹ made an extensive study of manganese-iron-carbon alloys about 1886, which culminated in the wide application and use of Hadfield austenitic manganese steel. His alloys were nearly all high in carbon, as compared with the steels discussed in this paper, and he found a marked tendency toward brittleness in the alloys of medium and low manganese content, that is, below 7 per cent. manganese. As a result, there has been a widespread prejudice against the use of manganese steels containing between 1.0 to 3.5 per cent. manganese with comparatively low carbon. Steels of this composition have never been studied adequately in comparison with other alloy steels, and have had little use in practice. In a very recent paper by Sir Robert Hadfield² the following statement appears: ". . . manganese-iron alloys with little or no carbon do not seem to offer any practical employment; they only become valuable in the presence of quite high percentages of carbon, 0.90 to 1.50 per cent."

The results of the present investigation are so contrary to the findings of Hadfield that the writers feel fully justified in presenting the preliminary report incorporated in this paper.

PREVIOUS INVESTIGATIONS

As early as 1886 the Terre Noire Co.* in France produced alloys containing up to 2.5 per cent. of manganese, using ordinary low-grade

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† Research Associate, Bureau of Metallurgical Research, Carnegie Institute of Technology.

‡ Assistant metallurgist, Pittsburgh Experiment Station, U. S. Bureau of Mines.

§ Mechanical engineer.

|| Sales engineer, Graham Bolt & Nut Co.

¹ R. A. Hadfield: Manganese in Its Application to Metallurgy and Some Newly Discovered Properties of Iron and Manganese. *Proc. Inst. Civil Eng.* (1887-88) 93, 61-76.

² R. A. Hadfield: Alloys of Iron and Manganese Containing Low Carbon. *Int. Iron Steel Inst.* (May, 1927) 65.

³ R. A. Hadfield: *Proc. British Inst. of Civil Eng.* (1888).

ferromanganese. Guillet⁴ made and studied low-carbon steels (0.05–0.40 per cent. carbon) containing manganese in amounts varying from 0.4 to 33.0 per cent., using carbon-free manganese produced by the Goldschmidt process. Arnold and Knowles,⁵ Lang,⁶ Abott,⁷ Matsushita,⁸ Kothny,⁹ Rawdon and Sillers,¹⁰ and Strauss,¹¹ have all given interesting results of tests on relatively low-carbon steels containing manganese as an alloy.

In America, Hall,^{12,13,14,15} and others, have given considerable attention to the use of steels with manganese between 1 and 2 per cent. and carbon below 0.3 per cent., especially for use in steel castings of high strength and toughness. Hall mentions the application of such steel in commercial castings as early as 1909.

Some additional references are to be found in the bibliography¹⁶ on manganese steels compiled by the Carnegie Library of Pittsburgh. The present investigation was undertaken by the U. S. Bureau of Mines at its Pittsburgh Experiment Station, jointly with the Carnegie Institute of Technology, the Walworth Co., and the Graham Bolt Nut Co. of Pittsburgh. Although the results to date can not be considered as final or conclusive, they include some interesting information which will be briefly summarized herein.

⁴ L. Guillet: *Aciers au Manganese. Bull. de la Soc. d'Encouragement pour l'Ind. Nationale* (1903) **105**, 421.

⁵ J. O. Arnold and F. K. Knowles: *The Mechanical Influence of Carbon on Alloys of Iron and Manganese. Jnl. Iron Steel Inst.* (1911) **85**, 76–85.

⁶ G. Lang: *The Influence of Manganese on the Physical Properties of Soft Steels. Metallurgie* (1911) **8**, 15–49.

⁷ R. R. Abott: *A Comparison of the Properties of a Nickel, Carbon and Manganese Steel before and after Heat Treatment. Trans. Am. Soc. Mech. Eng.* (1915) **37**, 39–46.

⁸ T. Matsushita: *Influence of Manganese on the Physical Properties of Carbon Steels. Sci. Repts., Tohoku Univ., Sendai, Japan* (No. 8, 1919) 79.

⁹ E. Kothny: *Stahl u. Eisen* (1919) **39**, 1341–1348.

¹⁰ H. S. Rawdon and F. Sillers, Jr.: *Effect of Manganese on Carbon Steels. Iron Age* (1922) **110**, 1357–1361.

¹¹ J. Strauss: *Characteristics of Some Manganese Steels. Trans. Am. Soc. Steel Tr.* (1923) **4**, 665–708.

¹² J. H. Hall: *Heat Treatment of Hypoeutectoid Carbon Steel Castings. Proc. Amer. Soc. Test. Mat.* (1913) **13**, 514.

¹³ J. H. Hall: *Shock Tests of Cast Steels. Trans.* (1913) **47**, 482–522.

¹⁴ A. E. Nissen, K. Taylor and J. H. Hall: *The Heat Treatment of Cast Steel. Trans.* (1920) **62**, 353.

¹⁵ J. H. Hall: *Pearlitic and Sorbitic Manganese Steels. Iron Age* (1922) **110**, 786–789.

¹⁶ *Manganese Steel*—A bibliography by E. H. McClelland and V. S. Polansky, with a preface by Sir Robert Hadfield, 60. Carnegie Library of Pittsburgh, 1926.

COMPOSITION OF STEELS STUDIED

As part of this investigation 36 heats of steel were produced in a 300-lb. electric furnace. These steels had a range of 0.07 to 0.35 per cent. carbon and 1.4 to 3.96 per cent. manganese, as shown in Fig. 1, the other elements being of the same order as found in carbon steels.

Since the term "manganese steels" is generally used to designate the austenitic steels with 12 to 15 per cent. manganese, and about 1.25 per cent. carbon, those treated in this paper will be called "low-carbon manganese."

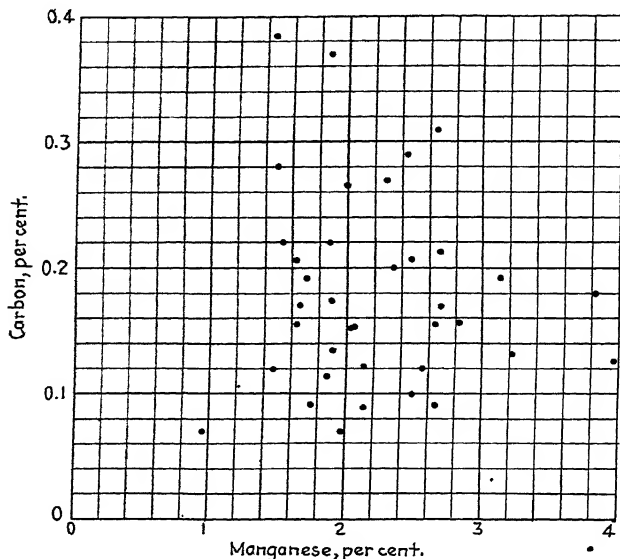


FIG. 1.—RANGE OF CARBON AND MANGANESE IN STEELS PRODUCED.

The effect of silicon on these steels has not been definitely determined, but it probably should be limited to 0.4 per cent. or less, especially in rolled or forged products.

A sulfur content as high as 0.153 per cent. occurring in some heats apparently had neither unfavorable nor favorable effect on the properties of the steel.

The only special feature involved in making these steels is the necessity of adding rather large quantities of manganese and at the same time keeping the carbon content relatively low. As manganese is removed from a bath of steel by oxidation more easily and rapidly than carbon, the alloy additions used to obtain the desired manganese content must not raise the carbon above that required in the finished steel. This can

be accomplished either by using silico-manganese (containing from 0.6 to 1.25 per cent. carbon) or special low-carbon ferromanganese alloys. In order to keep the silicon content low in the finished steel, silico-manganese should be added in the furnace to a wild or only partly deoxidized bath, so that most of the silicon will be oxidized out.

Manganese additions should, if possible, always be made in the furnace. Heats made with the manganese alloys added in the ladle have shown variations of as much as 0.50 per cent. in manganese and from 0.10 to 0.15 per cent. in carbon in samples taken from different parts of the

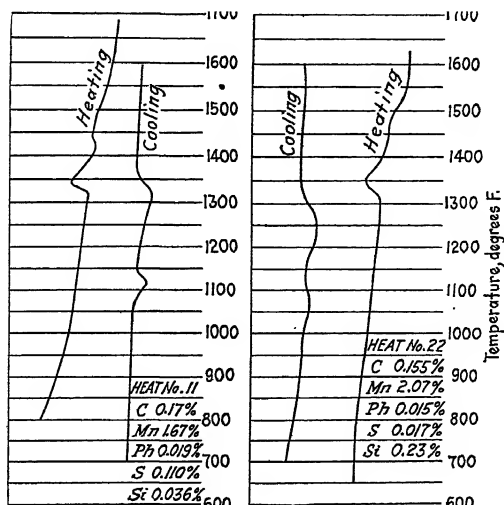


FIG. 2. HEAT No. 11, GROUP II.
FIG. 3. HEAT No. 22, GROUP II.

same heat. Heats finished in the ladle were not as clean nor as uniform as those made entirely in the furnace.

The practice in making these steels in the electric furnace is much the same as that in the open-hearth furnace, except that in the former the heat may be finished with a neutral or slightly reducing slag and the loss of manganese to the slag may be largely eliminated. After a heat is melted under an oxidizing slag and the carbon reduced to a value slightly below the final content desired, silico-manganese may be added to give a manganese content somewhat below that of the final analysis. The slag is then carefully deoxidized by sprinkling carbon over its surface. When the slag is neutralized, a sample for analysis is taken from the bath and the final additions of silico-manganese or ferromanganese are made and

the heat held from 10 to 20 min. before tapping. Small additions of ferrosilicon or aluminum may, of course, be made for final deoxidation.

In making small heats in the electric furnace it was found difficult to get consistent recoveries of manganese, probably because of a variable loss of manganese by vaporization. Approximate calculations¹⁷ from available data indicate that the vapor pressure of pure manganese at

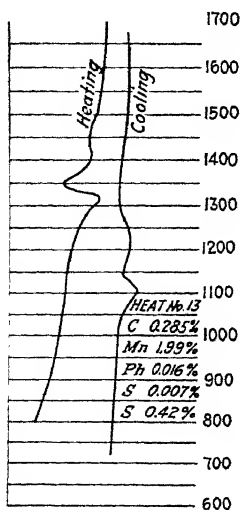


FIG. 4.—HEAT No. 13, GROUP III.

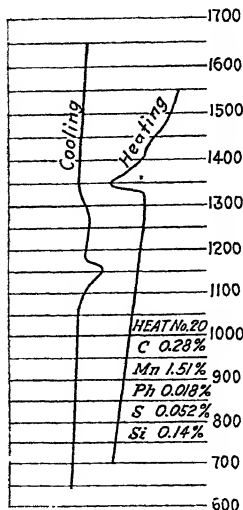


FIG. 5.—HEAT No. 20, GROUP III.

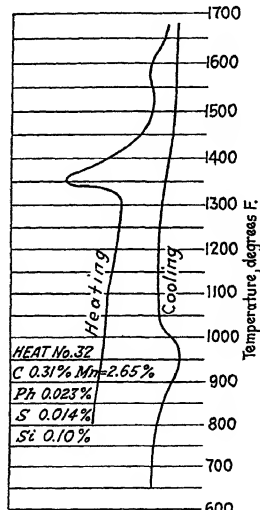


FIG. 6.—HEAT No. 32, GROUP III.

1600° C. is about 100 mm. of mercury, and if additions of manganese alloy are not quickly absorbed by the steel, or if they are melted under the arc, a large loss by vaporization seems possible. It is always preferable in making furnace additions to have the alloy in fairly large masses, so that it will sink through the slag and be quickly absorbed by the bath.

TRANSFORMATION POINTS OF LOW-CARBON MANGANESE STEELS

The determination of critical points in low-carbon manganese steels was obviously a natural starting point for this investigation. It was carried out on a hand-controlled Leeds and Northrup transformation-point apparatus with the following precautions: for each determination the rate of heating averaged 13° F. per min. and the rate of cooling 9°

¹⁷ J. Johnson: 'The Vapor Pressure and Volatility of Several High-boiling Metals—A Review. *Ind. & Eng. Chem.* (1917) 9, 873-878.

per min. The same sensitivity of the apparatus was maintained, except in special cases when higher sensitivity was desirable in order to verify results. The starting temperature of cooling was always 1800° F.

The temperature corresponding to the maximum deflection of the difference galvanometer was taken as the temperature of transformation. More than 75 critical curves were taken, most of them repeated two to four times. Although these experimental heats differ in so far as both carbon and manganese are concerned, no attempt is made either to describe in detail the critical points in each heat or to reproduce all the critical curves.

To simplify description, the steels are divided arbitrarily into several groups, the classification depending not only on carbon and manganese content, but also on the general appearance of critical curves.

Group I.—Carbon below 0.10 Per Cent. Manganese 1.75–2.75 Per Cent.

A₁ on heating 1350–1360° F.

A₁ on cooling 950–1130° F.

A₂₋₃ on heating 1480–1500° F.

A₂₋₂ on cooling 1200–1310° F.

The upper critical point is designated as A₂₋₃ on the assumption that the A₂ and A₃ points are merged. Critical transformation A₁ on heating and especially on cooling does not, apparently, take place in a narrow temperature range, but occupies a range of from 50 to 100 degrees, depending on the relative amounts of manganese and carbon. As the percentage of manganese increases the range of temperature occupied by the transformation becomes wider, while the intensity of transformation rapidly decreases. This indicates a sluggishness in the changes taking place at carbon point in this class of steels. It has been found that in a steel with 0.09 per cent. carbon and 2.63 per cent. manganese the temperature of the sample fell 120 degrees before A₁ transformation was completed.

Group II.—Carbon 0.15–0.20 Per Cent. Manganese 1.60–3.15 Per Cent.

Ac₁–1435°–1350° F.

Ar₁–950°–1100° F.

Ac₂₋₃–1450° F., or slightly higher

Ar₂₋₂–1200°–1300° F.

This group is characterized by somewhat higher carbon and the A₁ transformation is consequently more pronounced. It remains, however, quite sluggish. For instance, in steel with 0.155 per cent. carbon and 2.07 per cent. manganese (Fig. 3) Ac₁ point occupies a range of about 50 degrees, while Ar₁ is spread over 80 degrees, beginning at 1100° and ending at 1020°. Similarly Ar₂₋₂ covers 100 degrees with maximum transformation at 1205°. When manganese is increased to 3.14 per cent., Ac₂₋₃ point is very feeble. On cooling only one point could be detected (at about 950° F.).

Group III.—Carbon 0.25–0.31 Per Cent. Manganese 1.50–2.65 Per Cent.

Ac₁ 1350–1360° F.

Ac₂₋₃ is found to be at 1450° F.; in most cases it is quite feeble.

The temperatures at which transformations occur on cooling vary greatly, sometimes only one point being observed. For example, a steel with 0.31 per cent. carbon and 2.65 per cent. manganese may be cited: in this steel there was observed but one critical point at the temperature of 960° F.

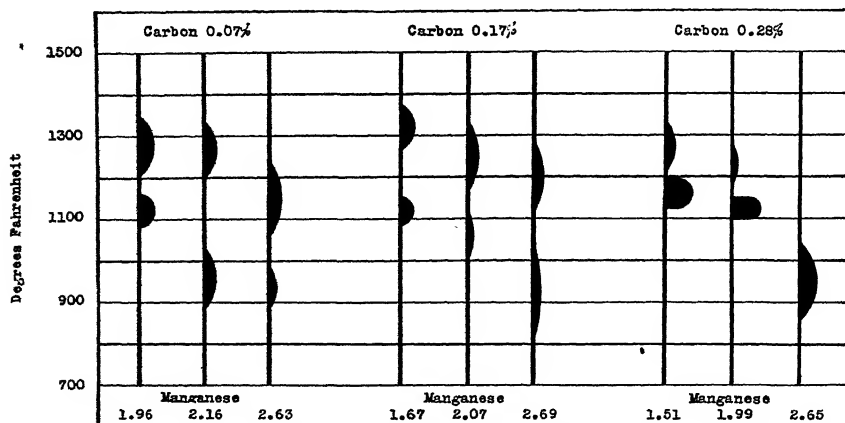


FIG. 7.—SUMMARY DIAGRAM OF CRITICAL POINTS.

In cases when only one point on cooling was found, or when the transformation was exceedingly slow (as it is termed here, "sluggish") higher sensitivity of the apparatus was employed in order to verify results.

In Table 1 are given the summarized results of a good many determinations.

TABLE 1.—Critical Points in Low-carbon Manganese Steels

Analysis, Per Cent.		Ac ₁ Deg. F.	Ac ₂₋₃ Deg. F.	Ar ₂₋₃ Deg. F.	Ar ₁ Deg. F.
C	Mn				
0.07-0.09	1.96	1360	1500	1310	1130
	2.16	1350	1480	1250	950
	2.63	1350	1480	1200	950
0.155-0.17	1.67	1345	1450	1320	1120
	2.07	1350	1480	1250	1050
	2.69	1350	1480	1200	950
0.19	3.14	1350	?	950?	
0.28-0.31	1.51	1360	1460	1270	1160
	1.99	1360	1450	1230	1120
	2.65	1350	1560?	960	

The same table is diagrammatically represented in Fig. 7. The width of black areas represents the intensity of transformation, while the length gives an idea of the range of temperatures covered during transformation.

PHYSICAL PROPERTIES OF LOW-CARBON MANGANESE STEEL

The task of studying the physical properties of any group of steels is so great that the authors should be pardoned for the sketchy account given in this paper.

Range of Properties Developed by Heat Treatment

Preliminary work on two heats of the composition given in Table 2 indicated that a wide range of properties can be developed. At the same time, as further study proved, the low-carbon manganese steels are not "tender" nor sensitive to small variations in heat treatment. It became apparent almost from the beginning that mere annealing or normalizing did not develop the best qualities in this type of steels.

Heat treatment, consisting of quenching in water from temperatures well above the critical range followed by a draw at various temperatures indicated better possibilities for the development of interesting qualities in this type of steels.

In Table 2 are given a few figures to enable us to draw a comparison between the steels "as rolled" and after heat treating.

Table 3 shows the tensile test results for samples from various heats. These are given to convey some idea of the maximum values for tensile strength, proportional limit, and ductility, which can be developed in this group of steels by a simple heat treatment, consisting of a quench in water followed by a draw between 600° and 1050° F.

The maximum elongation in 2 in. is about 30 per cent., with as high as 75 per cent. reduction in area. The highest values for proportional limits are about 145,000 lb. per sq. in. Several steels drawn at 600° F. with manganese content between 2 and 3 per cent. and carbon 0.20 to 0.30 per cent. gave tensile strengths above 200,000 lb. with elongation about 10 per cent. and reduction of area of 30 per cent. All these pieces could be readily machined and drilled in the heat-treated condition.

Since the available data, though numerous, are widely scattered, it is not possible to indicate definitely either the best heat treatment or any quantitative relation between composition and physical properties. There are reasons to believe that a double heat treatment, including a water quench from rather high temperature, is necessary to develop the best properties in these steels. For this purpose Table 4 is offered for consideration. It contains numerous data relative to the properties developed in low-carbon manganese steels of different composition after various heat treatments. Careful study of this table indicates at once

the importance of proper heat treatment. Steel of 0.215 per cent. carbon and 2.69 per cent. manganese is included in the table to illustrate this point. By varying the quenching temperature and subjecting the steel to double treatment, as described in the table, the elastic limit is raised from 55,000 to 139,000 lb. per sq. in., while the ductility of the steel remains practically unchanged.

TABLE 2.—*Indication of Range of Physical Properties Developed by Heat Treatment in Low-carbon Manganese Steels*

Heat	C, Per Cent.	Mn, Per Cent.	Heat Treatment	Proportional Elas. Limit Lb. per Sq. in.	Ultimate Strength, Lb. per Sq. in.	Elong. in 2 In., Per Cent.	Reduction in Area, Per Cent.
A	0.27	1.97	As rolled.	50,180	81,200	22.0	47.0
	0.27	1.97	Water-quenched at 1600°, draw 2 hr. at 1050°.	90,900	115,600	21.0	61.1
	0.27	1.97	Water-quenched at 1600°, draw 2 hr. at 800°.	143,500	169,200	11.5	36.6
	0.27	1.97	Water-quenched at 1700°, draw 2 hr. at 600°.	134,300	203,200	6.5	21.1
B	0.21	3.18	As rolled.	81,200	121,600	4.0	21.8
	0.21	3.18	Water-quenched at 1900°, draw 2 hr. at 800°.	130,300	165,200	12.0	48.4

TABLE 3.—*Range of Tensile Test Properties Developed by Heat Treatment*

Mark	Composition, Per Cent.			Heat Treatment		Proportional Elas. Limit, Lb. per Sq. in.	Ultimate Strength, Lb. per Sq. in.	Elong. in 2 In., Per Cent.	Reduction in Area, Per Cent.
	C	Mn	Si	Water Quench after ½ Hr. at Degrees F.	Draw for 2 Hr. at Degrees F.				
16-6	0.12	2.16	0.29	1,600	1,050	73,400	91,400	29.5	70.8
10-6	0.155	1.64	0.07	1,600	1,050	65,400	85,080	25.5	72.5
8-3	0.12	1.46	0.51	1,700	1,050	72,150	95,400	28.0	69.0
8-3	0.205	1.64	0.44	1,700	1,050	55,700	73,400	29.0	75.7
36-5	0.275	1.97	0.10	1,600	800	143,500	169,200	11.5	36.6
27-5	0.22	2.50	0.38	1,600	800	147,300	171,000	11.5	43.6
27-8	0.22	2.50	0.38	1,800	800	146,000	179,200	12.5	41.1
18-11	0.20	2.38	0.13	1,900	800	138,300	170,500	14.0	52.6
7-4	0.29	2.43	0.45	1,600	600	130,000	202,200	8.5	26.1
36-1	0.275	1.97	0.10	1,700	600	134,300	203,200	6.5	21.1
27-7	0.22	2.50	0.38	1,800	600	134,000	200,700	10.0	35.6
7-10	0.29	2.43	0.45	1,900	600	125,000	209,500	12.5	43.6

TABLE 4.—Data Regarding Properties Developed in Low-carbon Manganese Steels by Heat Treatment

Line No.	Heat No.	Carbon	Manganese	Heat Treatment	Elastic Limit, Lb. per Sq. in.	Yield Point, Lb. per Sq. in.	Ultimate Strength, Lb. per Sq. in.	Elongation in 2 in., Per Cent.	Reduction in Area, Per Cent.	Remarks
1	28	0.09	2.16	As rolled.	40,600	88,980	103,800	20.0	44.7	
2	36B	.275	1.97	As rolled.	50,180	81,200	97,640	22.0	47.0	
3	15	.12	2.58	As rolled.	63,920	91,980	104,000	20.0	40.2	
4	28B	.215	3.18	As rolled.	81,200	121,300	127,600	4.0	21.8	Fracture coarse grain
5	10	.115	1.88	Heated 1550° F. $\frac{1}{2}$ hr. Water-quenched, drawn at 1000° F. for 2 hr.	51,940	91,860	97,860	23.0	66.6	
6	9	.07	1.96		69,640	84,120	94,600	20.0	69.4	
7	11	.17	1.67		71,900	89,120	95,360	20.0	53.1	
8	6	.12	1.46		79,880	96,350	104,300	24.0	63.1	
9	12	.155	2.64		93,860	108,800	109,800	20.0	60.4	
10	4	.17	2.69		96,860	110,400	115,200	15.0	41.4	
11	5	.27	2.30		112,300	124,800	131,300	20.5	68.2	
12	9	.07	1.96	Same as above except drawn at 800° F. for 2 hr.	74,380	105,300	114,300	22.0	60.9	
13	11	.17	1.67		77,880	105,300	112,300	15.5	43.7	
14	6	.12	1.46		90,800	123,300	138,300	17.0	54.0	
15	10	.115	1.88		94,360	123,800	127,300	13.0	43.4	
16	12	.155	2.64		125,000	155,750	157,300	8.0	29.2	
17	4	.17	2.69		137,300	179,000	180,200	7.0	23.5	
18	5	.27	2.30		153,420	179,000	187,600	9.0	29.2	
19	10	.115	1.88	Same as above except drawn at 600° F. for 2 hr.	70,400	117,300	121,300	15.0	48.4	
20	9	.07	1.96		71,680	109,500	118,800	22.0	52.1	
21	11	.17	1.67		86,870	121,800	124,100	9.0	23.1	
22	6	.12	1.46		90,230	127,300	130,300	8.0	35.9	
23	12	.155	2.64		118,300	168,000	168,800	7.5	21.3	
24	5	.27	2.30		168,300	221,200	221,200	3.5	1.2	
25	4	.17	2.69		169,500	188,000	189,700	2.5	22.6	
26	14	.215	2.69	Quench temperature 1650° F., 3 hr. air	55,420	92,360	103,600	19.0	25.4	
27	14	.215	2.69	QT 1800° F., 1 hr. air.	70,150	94,080	104,500	14.0	14.4	
28	14	.215	2.69	QT 1650° F., $\frac{1}{2}$ hr. water, drawn 1000° F. 2 hr.	86,960	96,240	108,600	22.0	61.7	
29	14	.215	2.69	QT 1400° F., $\frac{1}{2}$ hr. water, drawn 1000° F. 2 hr.	92,360	96,340	108,100	24.0	58.9	
30	14	.215	2.69	Double treatment 1720° F., 8 hr. water, 1600° F. $\frac{1}{2}$ hr. water, drawn 800° F. 9 hr.	139,300	157,400	159,400	12.5	50.7	

TABLE 5.—*Variation in Physical Properties with Quenching Temperatures*

Mark	C, Per Cent.	Mn, Per Cent.	Heat Treatment	Proportional Elas. Limit, Lb. per Sq. In.	Ultimate Strength, Lb. per Sq. In.	Elong. in 2 In., Per Cent.	Reduction in Area, Per Cent.
C	0.115	1.94	Water-quenched $\frac{1}{2}$ hr. at 1600° F., drawn 2 hr. at 800° F.	67,900	97,100	17.5	53.6
	0.115	1.94	Water-quenched $\frac{1}{2}$ hr. at 1700° F., drawn 2 hr. at 800° F.	64,900	95,300	16.5	55.1
	0.115	1.94	Water-quenched $\frac{1}{2}$ hr. at 1800° F., drawn 2 hr. at 800° F.	67,650	96,850	19.0	59.1
	0.115	1.94	Water-quenched $\frac{1}{2}$ hr. at 1900° F., drawn 2 hr. at 800° F.	74,300	112,300	17.5	54.1
D	0.22	2.50	Water-quenched $\frac{1}{2}$ hr. at 1600° F., drawn 2 hr. at 800° F.	147,300	171,000	11.5	43.6
	0.22	2.50	Water-quenched $\frac{1}{2}$ hr. at 1800° F., drawn 2 hr. at 800° F.	146,000	179,200	12.5	41.1
	0.22	2.50	Water-quenched $\frac{1}{2}$ hr. at 1900° F., drawn 2 hr. at 800° F.	144,800	178,200	11.0	43.1

TABLE 6.—*Variation in Physical Properties with Quenching Temperatures*

Mark	C, Per Cent.	Mn, Per Cent.	Heat Treatment	Proportional Elas. Limit, Lb. per Sq. In.	Ultimate Strength, Lb. per Sq. In.	Elong. in 2 In., Per Cent.	Reduction in Area, Per Cent.
E	0.155	1.64	Water-quenched $\frac{1}{2}$ hr. at 1600° F., drawn 2 hr. at 800°		105,400	17.0	62.4
	0.155	1.64	Water-quenched $\frac{1}{2}$ hr. at 1700° F., drawn 2 hr. at 800°	91,000	102,100	17.5	64.3
	0.155	1.64	Water-quenched $\frac{1}{2}$ hr. at 1800° F., drawn 2 hr. at 800°	98,800	128,800	10.0	34.7
	0.155	1.64	Water-quenched $\frac{1}{2}$ hr. at 1900° F., drawn 2 hr. at 800°	97,800	116,100	11.5	44.9
F	0.28	1.51	Water-quenched $\frac{1}{2}$ hr. at 1600° F., drawn 2 hr. at 800°	97,100	127,300	16.5	39.4
	0.28	1.51	Water-quenched $\frac{1}{2}$ hr. at 1700° F., drawn 2 hr. at 800°	87,900	122,100	16.5	67.8
	0.28	1.51	Water-quenched $\frac{1}{2}$ hr. at 1800° F., drawn 2 hr. at 800°	120,000?	147,300	5.0	19.2
	0.28	1.51	Water-quenched $\frac{1}{2}$ hr. at 1900° F., drawn 2 hr. at 800°		177,700	2.5	5.5

While inviting attention to some mechanical properties of this type of steel, the authors of this paper do not wish to make any definite or excessive claims as to its superiority over other steels. As mentioned, the present study is an attempt to fill up a gap in the knowledge of steels with manganese as an alloying element. The results as presented herein are considered preliminary and the work far from complete.

It is believed that interesting results might reasonably be expected from steels higher in manganese and lower in carbon, especially in regard to shock and fatigue-resisting qualities, and the work at the present time is confined to the study of steel of this character.

Results Obtained on Water-quench from Various Temperatures

A special study of the different temperatures of quenching is presented in Tables 5 and 6, which include representative results obtained on four steels of various compositions after quenching in water from temperatures of 1600° up to 1900° F., followed by a draw for 2 hr. at 800° F. Certain observations can be made from the study of these two tables.

The first observation made was that some steels of a certain carbon and manganese content (steel C, Table 5) do not show any appreciable change in ductility, even though heating before quenching is carried well above the critical range. It will be noted that as the quenching temperature is raised the proportional limit and ultimate strength are gradually increased, but this increase in strength is not accompanied by the loss of ductility.

As the carbon and manganese are increased the raising of quenching temperature seems to have very little effect on the mechanical properties, which, as a matter of fact (steel D, Table 5) remain practically identical.

This, however, does not hold true for steels with larger ratios of carbon to manganese (steels E and F, Table 6) which are similar to carbon steels in that they suffer the loss of ductility as the strength is increased.

Further increase in quenching temperatures did not seem to have any detrimental effect on the mechanical properties of steel. On the contrary, as will be shown later, some of the best mechanical tests were obtained using high initial quenching temperature.

Effect of Varied Drawing Temperatures

Figs. 8 and 9 give the results obtained by varying draw temperatures on bars quenched in water from 1600° to 1900° F. Pieces from each quenching treatment were given a 2-hr. draw at 600°, 800°, and 1050°. As would be expected, the ultimate tensile strength decreases and ductility increases with rising draw temperatures. However, all proportional-limit curves show a maximum which appears to be near the 800° F. draw temperature. The difference between the proportional limit and the

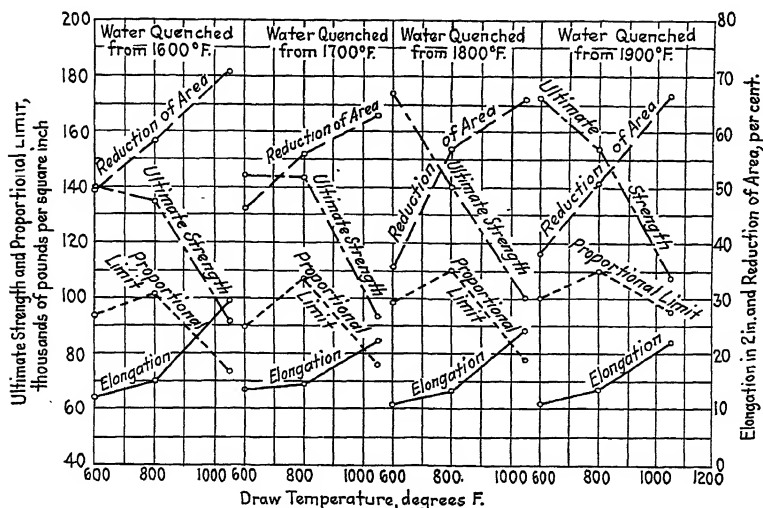


FIG. 8.—RESULTS OBTAINED BY VARYING DRAW TEMPERATURES ON BARS QUENCHED IN WATER FROM 1600° TO 1900° F. ANALYSIS: CARBON 0.22 PER CENT., MANGANESE 2.16 PER CENT., SILICON 0.29 PER CENT.

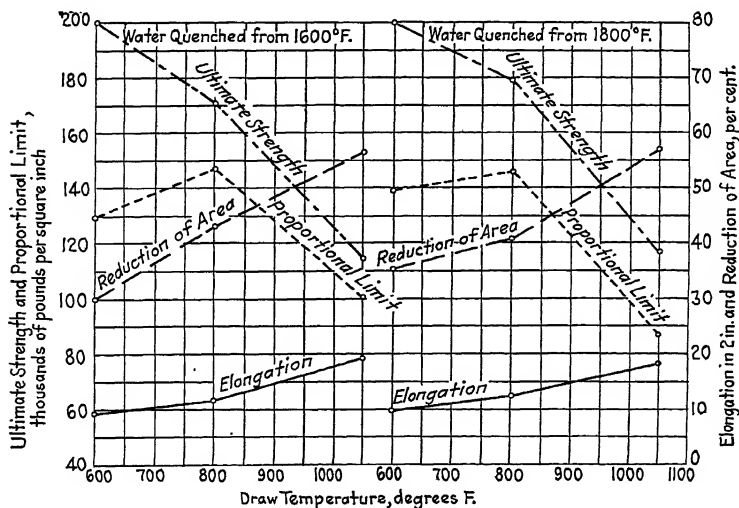


FIG. 9.—RESULTS OBTAINED BY VARYING DRAW TEMPERATURES ON BARS QUENCHED IN WATER FROM 1600° TO 1900° F. ANALYSIS: CARBON 0.22 PER CENT., MANGANESE 2.5 PER CENT., SILICON 0.38 PER CENT., SULFUR 0.12 PER CENT.

ultimate tensile strength decreases sharply as the drawing temperature is increased and then more slowly as the draw is carried to still higher temperatures.

STRESS-STRAIN CURVES

During the testing of differently treated steels, it was observed that the yield point as commonly defined was so far above the true proportional limit that its observation was quite difficult. The numerical value of yield point could be ascertained only if the beam of the test machine was carefully watched and the load applied quite slowly. Even under

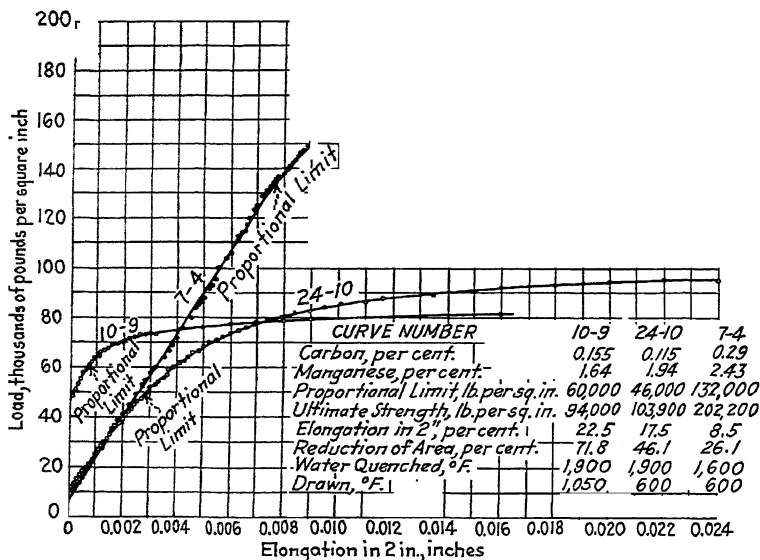


FIG. 10.—STRESS-STRAIN CURVES FOR LOW-CARBON MANGANESE STEELS.

these conditions the drop of the beam was very slight. In the early work on this material the "yield point" values were reported, but as work progressed the proportional limit was determined from the stress-strain diagrams and reported in place of yield point. The difference in the values of the "proportional limit" and the "yield point" is unusually large, as is shown in columns 4 and 5 of Table 4. It is believed that in steels where the carbon is relatively high and manganese relatively low—that is, in steels that depend on carbon rather than manganese for their chief strengthening element—the yield point is quite pronounced, but as the manganese is raised to a point where it becomes the dominating strengthening factor, the yield point loses its distinctness and may be easily misjudged.

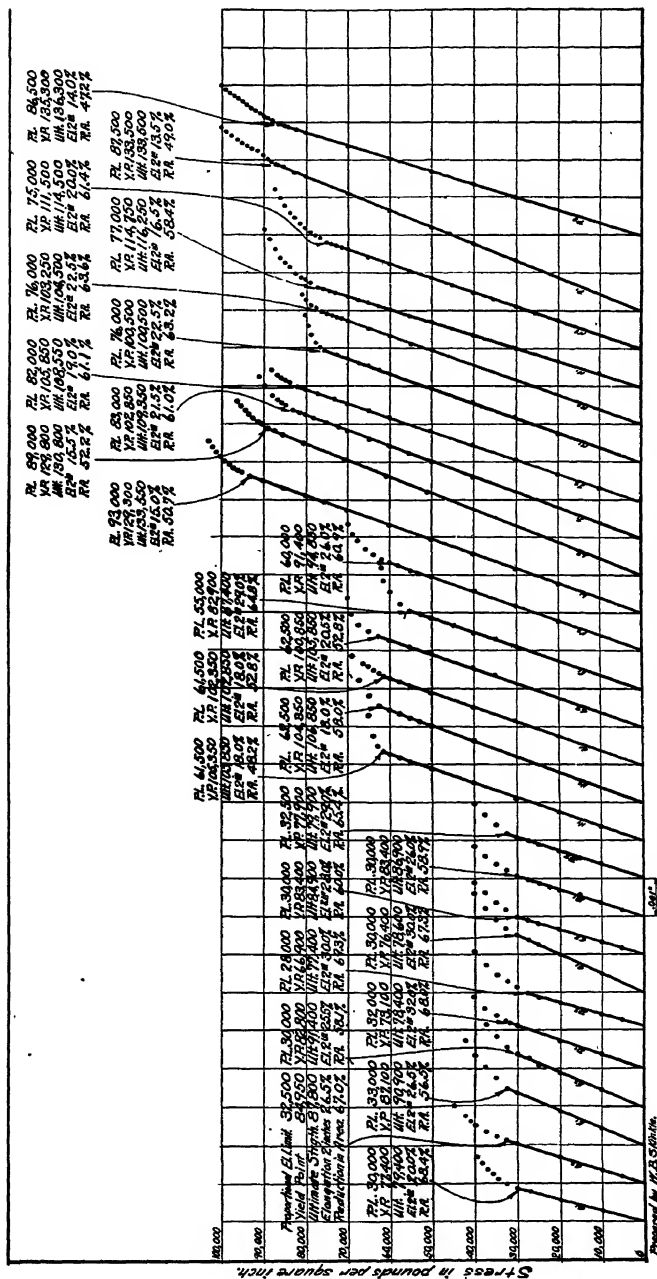


FIG. 11.—COMPARISON OF STRESS-STRAIN CURVES.

Approved by W. B. Spillie.

Fig. 10 shows a few typical stress-strain curves at room temperatures for these steels in a heat-treated condition. In nearly all cases there is no break in the curves above the proportional limit, but only a gradual sloping off to the ultimate stress values. The curves shown in this figure illustrate rather well the point brought up for discussion; namely, that these steels gave no true yield point. Fig. 11 represents a graphical study of the mechanical properties developed in 0.17 per cent. carbon, 2.07 per cent. manganese steel, when the latter is subjected to different heat treatments. The first ten curves (*A* to *E* inclusive) were obtained by heating the steel to a temperature between 1600°–1900° F., holding for 1 hr. and cooling in air. The values obtained for mechanical properties are not exceptional.

By changing the treatment to one consisting of normalizing in air, followed by a water quench from 1350°–1500° F. and a draw between 1000° and 800° F., the values, as can be seen from curves *G–F* inclusive, for proportional limit and ultimate strength are much increased, while the ductility remains very good.

Curves *L* and *J* show the results of the following heat treatment: normalized from 1900° F., reheated to 1500° F., rapidly cooled to 1350° F., quenched in water and drawn between 800°–1000° F.

Finally, the last six curves (*M*, *N* and *P*) represent a quench in water from a high temperature, followed by another, also in water, from 1400° F. and a draw at the same temperature as before. These last values are rather interesting in view of excellent ductility combined with good strength.

Fig. 12 gives some preliminary results on the behavior of these steels at high temperatures.

IMPACT TESTS

Langenberg¹⁸ reports results of Charpy impact tests on a series of cast steels as shown in Table 7.

TABLE 7.—*Results of Charpy Impact Tests on Cast Steels (Langenberg)*

No.	Carbon, Per Cent.	Manganese, Per Cent.	Charpy Impact Value, Ft.-lb., Average
1	0.44	0.93	10.01
2	0.27	1.08	31.58
3	0.298	1.31	32.24
4	0.222	1.29	27.87
5	0.33	1.54	8.49
6	0.346	1.76	6.75

¹⁸ F. C. Langenberg: Impact Tests of Cast Steel. *Proc. Amer. Soc. Test. Mat.* (1921) **21**, 840–851.

These indicate that the toughness, as measured by shock resistance, drops rapidly with a carbon content above 0.30 per cent., even with higher manganese content.

Table 8 gives the results of tests made in the course of the present study. A plain carbon steel (carbon 0.30 per cent.; manganese 0.50 per cent.) was compared with a manganese steel (carbon 0.25 per cent.; manganese 1.75 per cent.), in both tensile and Izod impact tests. The carbon steel was oil-quenched from 1475°, withdrawn from the oil bath at between 200° and 400° F. and cooled in air. We were advised that this treatment produces the maximum impact toughness for this type of steel. The manganese steel was quenched in water from about 1800° F. and drawn at 800°, 900° and 1000° for 2 hr. One tensile test-bar and six impact test-bars were tested from each heat treatment. The carbon steel shows values for elongation and tensile strength fairly comparable to the average of values for samples of manganese steel, but the resistance to impact is noticeably lower and more irregular in the carbon steel. From these and other results not yet compiled for publication, it seems that a certain kind of toughness caused by the presence of manganese is revealed by

TABLE 8.—*Izod Impact Tests on Low-carbon Manganese Steel Compared with a Carbon Steel*

Composition, Per Cent.		Heat Treatment	Elastic Limit, Lb. per Sq. In.	Yield Point, Lb. per Sq. In.	Ultimate Strength, Lb. per Sq. In.	Elong. in 2 In., Per Cent.	Reduction in Area, Per Cent.	Izod Impact Ft.-lb. to Destruction	Remarks
C	Mn								
0.28	0.50	At 1475° for ½ hour; oil-quenched; withdrawn from oil 200° to 400°; cooled in air.	75,000	106,000	107,000	14.5	33.5	14	Average of tests, 7.16 ft.-lb.
								9	
								9	
								4	
0.25	1.75	Water-quenched from 1775° to 1850°; drawn at 800° for 2 hr.	104,000	132,000	133,000	10.0	39.5	4	Average of tests, 21.5 ft.-lb.
								3	
								22	
								21	
0.25	1.75	Same as above except drawn at 900° for 2 hr.	96,000	118,000	119,000	13.0	52.0	22	Average of tests, 36.5 ft.-lb.
								22	
								34	
								34	
0.25	1.75	Same as above except drawn at 1000° for 2 hr.	71,000	86,000	89,000	25.5	68.4	41	Average of tests, 61.4 ft.-lb.
								37	
								39	
								61	
								61	
								62	
								64	
								69*	
								59	

* Defective notch.

impact values better than by any other kind of physical test and the subject deserves much further study.

FATIGUE TESTS

The authors are not aware of any comprehensive work done in relation to the fatigue properties of steels low in carbon and with manganese between 1.75-3.0 per cent. Moore¹⁹ reports results of fatigue tests on two cast steels, however, one containing 0.25 per cent. carbon and 0.68 per cent. manganese, and the other 0.35 per cent. carbon and 1.71 per cent. manganese. The endurance ratio (endurance limit divided by ultimate tensile strength) was 0.43 for both steels. The endurance limits after various heat treatments averaged 33,000 lb. per sq. in. for carbon steel and 45,000 lb. per sq. in. for manganese steel. A limited number of fatigue tests were made by F. F. McIntosh, formerly professor of metallurgy at the Carnegie Institute of Technology. A 0.50 per cent. carbon steel was compared with one containing 0.25 per cent

TABLE 9.—*Fatigue Tests on a Low-carbon, Manganese Steel and a Plain Carbon Steel*

(Both samples cut from high-grade deep-well drill rod)

Tensile Test

Mark	Composition, Per Cent.		Yield Point, Lb. per Sq. In.	Ultimate Strength, Lb. per Sq. In.	Elonga- tion in 2 In., Per Cent.	Reduction in Area, Per Cent.	Remarks
	C	Mn					
C	0.50		59,200	88,300	27.3	44.6	As rolled.
C	0.50		47,815	86,120	23.9	37.0	As rolled.
M	0.25	1.78	57,325	90,445	30.4	61.0	As rolled.
M	0.25	1.78	56,430	89,043	32.8	65.0	As rolled.
M	0.25	1.78	55,280	90,630	28.1	44.7	Annealed.
M	0.25	1.78	57,150	89,750	31.2	49.0	Annealed.

Fatigue Tests

Mark	Composition, Per Cent.		Stress in Test Specimen, Lb. per Sq. In.	Revolutions of Test Ma- chine before Failure of Specimen	Remarks
	C	Mn			
C	0.50		40,000	20,000,000	No failure.
M	0.25	1.78	40,000	20,000,000	No failure.
C	0.50		45,000	208,740	Ratio: 1:2.6.
M	0.25	1.78	45,000	543,120	
C	0.50		50,000	69,100	Ratio: 1:6.0.
M	0.25	1.78	50,000	414,000	

¹⁹ H. F. Moore: Tests of the Fatigue Strengths of Cast Steel, conducted by the Engineering Experiment Station, University of Illinois, in cooperation with the American Steel Foundries. *Bull.* 156, Univ. of Ill. Eng. Exp. Station (July, 1926) 23.

carbon and 1.78 per cent. manganese in a rotating-beam type of fatigue-test machine. The results are included in Table 9, which also gives the values for tensile tests on the same steels. At 45,000 lb. per sq. in. stress, the respective number of reversals before failure were in the ratio of 1.0 for the carbon steel to 2.6 for the manganese steel; at 50,000 lb. the ratio was 1.0 to 6.0.

METALLOGRAPHIC INVESTIGATION

With the wide variations in the heat treatment of steel it seemed advisable to follow the mechanical testing by metallographic study of the structures and, consequently, a voluminous number of photomicrographs is at the authors' command. It would be impractical to reproduce all of them but a few of the most representative or interesting are included, and these will permit us to follow each step in the heat treatment.



FIG. 13.—STEEL CONTAINING 0.17 PER CENT. C, 2.07 PER CENT. MN. $\times 150$. ETCHED WITH HNO_3 .

Reduced to approximately $\frac{3}{4}$ original size.

FIG. 14.—STEEL CONTAINING 0.17 PER CENT. C, 2.07 PER CENT. MN. HEATED TO 1800° F. FOR 1 HR.; AIR-COOLED. $\times 150$. ETCHED WITH HNO_3 .

FIG. 15.—STEEL CONTAINING 0.17 PER CENT. C, 2.07 PER CENT. MN. QUENCHED IN WATER FROM 1800° F. $\times 150$. ETCHED WITH HNO_3 .

FIG. 16.—STEEL CONTAINING 0.17 PER CENT. C, 2.07 PER CENT. MN. HEATED TO A HIGH TEMPERATURE, AIR-COOLED, REHEATED TO 1500° F., QUENCHED IN WATER, DRAWN AT 900° F. $\times 150$. ETCHED WITH HNO_3 .

FIG. 17.—STEEL CONTAINING 0.17 PER CENT. C, 2.07 PER CENT. MN. WATER-QUENCHED FROM A VERY HIGH TEMPERATURE, REHEATED TO 1400° F., WATER-QUENCHED, DRAWN AT 900° F. $\times 150$. ETCHED WITH HNO_3 .

FIG. 18.—STEEL CONTAINING 0.17 PER CENT. C, 2.07 PER CENT. MN. TREATMENT SAME AS FIG. 17.

FIG. 19.—STEEL CONTAINING 0.17 PER CENT. C, 2.07 PER CENT. MN. THE STRUCTURE OF LARGE PSEUDO-MARTENSITIC GRAINS OBTAINED FROM QUENCHING AT A VERY HIGH TEMPERATURE. $\times 150$. ETCHED WITH HNO_3 .

Reduced to approximately $\frac{3}{4}$ original size.

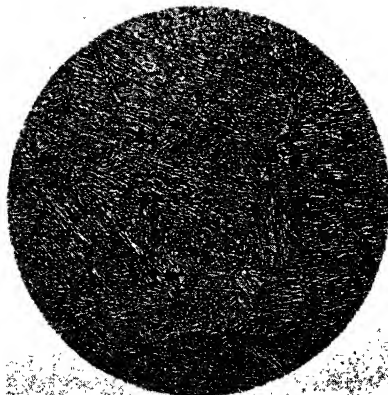
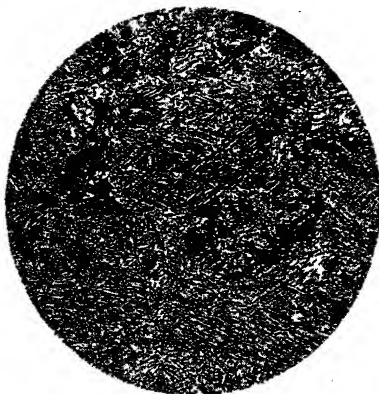
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17



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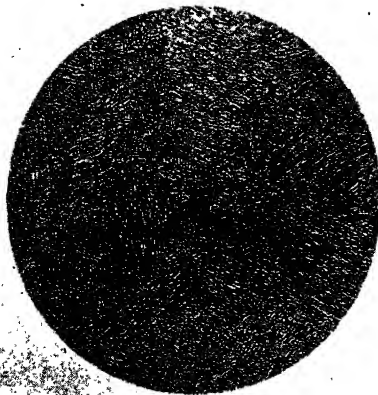


Fig. 13 shows 0.17 per cent. carbon, 2.07 per cent. manganese steel in "as rolled" condition. If annealed or normalized—for instance, by heating to 1800° F., holding for 1 hr. and air cooling—the structure (Fig. 14) becomes quite uniform and consists of sorbitic and ferritic grains, characteristic for all analyses within the scope of the paper. On quenching in water, oil or air, the structure is martensitic (Fig. 15). This martensitic structure is quite stable and on drawing undergoes only partial decomposition. As an example can be shown a steel with 0.17 per cent. carbon and 2.07 per cent. manganese, heated to a high temperature, air-cooled and then given second quench (in water) from about 1500° F. followed in turn by a draw at 900° F., which gives us a pseudo-martensitic structure with numerous troostitic grains (Fig. 16). It will be remembered (see Curve *G*, Fig. 11) that the mechanical properties after the treatment as described were much improved.

Giving still another treatment to the same steel—namely, water quench from a very high temperature, followed by a moderate quench in water from 1400° F. and draw at 900° F.—an interesting structure is obtained (Fig. 17).

The pseudo-martensitic formations are arranged in regular patterns and resemble the Widmanstätten structure of cast steels. It was noted that this structure is characteristic of the high initial quenching. At times, instead of forming troostitic patches (on drawing) the pseudo-martensitic formations become more sharply defined and less martensitic, yet preserve the characteristic appearance of the original quench structure (Fig. 18). Sometimes, if the original quenching temperature was quite high, a structure of large pseudo-martensitic grains can be obtained (Fig. 19). This structure was taken from the sample *P* (Fig. 11) and gave good mechanical properties.

During the study of heat treatment and the resultant structures, it was also observed that certain steels possessed rather unusual metallographic characteristics and considerable time was devoted to their investigation.

FIG. 20.—STEEL CONTAINING 0.12 PER CENT. C, 1.46 PER CENT. MN, "AS CAST."
× 150. ETCHED WITH HNO₃.

FIG. 21.—STEEL CONTAINING 0.12 PER CENT. C, 2.58 PER CENT. MN, "AS CAST."
× 150. ETCHED WITH HNO₃. (C. H. CHOU.)

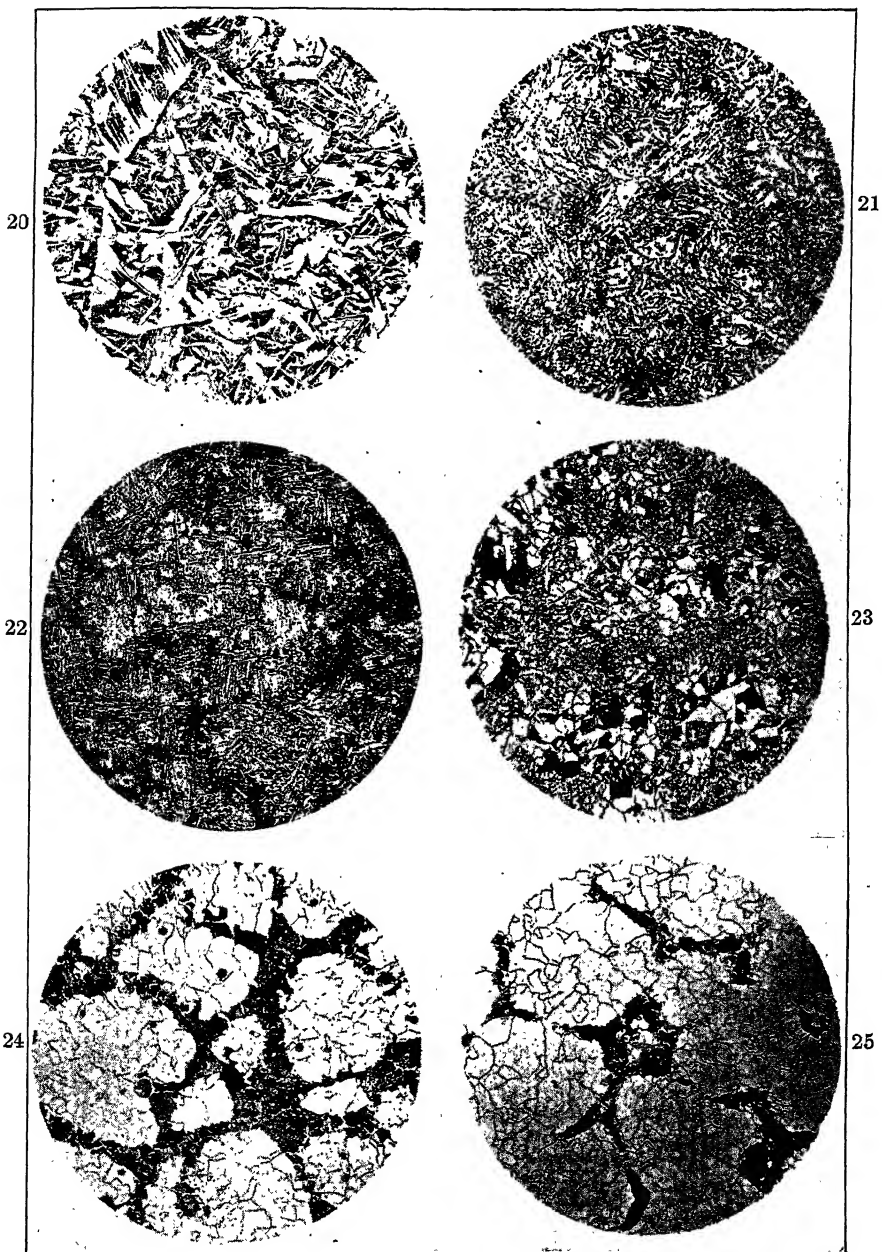
FIG. 22.—STEEL CONTAINING 0.31 PER CENT. C, 2.65 PER CENT. MN, "AS CAST."
× 150. ETCHED WITH HNO₃. (C. H. CHOU.)

FIG. 23.—STEEL CONTAINING 0.31 PER CENT. C, 2.65 PER CENT. MN, ANNEALED AT 1850° F. × 150. ETCHED WITH HNO₃. (C. H. CHOU.)

FIG. 24.—STEEL CONTAINING 0.12 PER CENT. C, 1.5 PER CENT. MN, ANNEALED AT 1850° F. FOR 1 HR. × 150. ETCHED WITH HNO₃. (C. H. CHOU.)

FIG. 25.—STEEL CONTAINING 0.12 PER CENT. C, 1.5 PER CENT. MN, ANNEALED AT 1850° F. FOR 2 HR., FURNACE-COOLED. × 150. ETCHED WITH HNO₃. (C. H. CHOU.)

Reduced to approximately $\frac{3}{8}$ original size.



It has been reported by one of the authors²⁰ that in alloys of iron and manganese the segregation of manganese due to selective solidification can be detected and that apparently manganese plays the same role in "dendritic segregation" as phosphorus, carbon and other elements. Furthermore, if even small quantities of carbon are present, the so-called "dendritic pattern" is much more pronounced.

Those who have worked with high-manganese steels, such as the well-known Hadfield manganese steels, are well aware that the dendritic segregation and resultant uneven distribution of carbides in these steels are difficult to break up. It seemed of interest, therefore, to examine a few cast bars of low-carbon manganese steel for their response to straight annealing and to ascertain whether their cast structure can be easily broken up.

Fig. 20 shows the structure of steel "as cast" of the average analysis of 0.12 per cent. carbon and 1.46 per cent. manganese. It presents the usual structure of cast steel, in which ferritic and sorbitic grains are definitely oriented, forming the so-called Widmanstätten structure. Figs. 21 and 22 show two other steels, one containing 0.12 per cent. carbon and 2.58 per cent. manganese and the other containing 0.31 per cent. carbon and 2.65 per cent. manganese. It will be observed that with the increase in manganese, even though the carbon remains the same, the structures of the steels "as cast" are quite different in appearance.

Fig. 21 shows an almost complete admixture of ferrite and cementite, with disseminated areas of either troostitic or sorbitic nature. It is interesting to note that dark sorbitic areas give an impression of a thick network. Fig. 22 is different in appearance and shows a large number of elongated particles which may be easily mistaken for martensitic needles; such a structure is sometimes called pseudo-martensitic. These elongated particles are embedded in the ferritic mass and are accompanied by small areas of sorbite.

²⁰ V. N. Krivobok: The Crystallization of Iron-carbon Alloys, 75. Thesis for degree of Sc. D., Harvard University, 1924.

FIG. 26.—STEEL CONTAINING 0.12 PER CENT. C, 1.46 PER CENT. MN, HOT ROLLED. $\times 150$. ETCHED WITH HNO_3 .

FIG. 27.—STEEL CONTAINING 0.12 PER CENT. C, 2.58 PER CENT. MN, HOT ROLLED. $\times 150$. ETCHED WITH HNO_3 . (C. H. CHOU.)

FIG. 28.—STEEL CONTAINING 0.31 PER CENT. C, 2.65 PER CENT. MN, HOT ROLLED. $\times 150$. ETCHED WITH HNO_3 . (C. H. CHOU.)

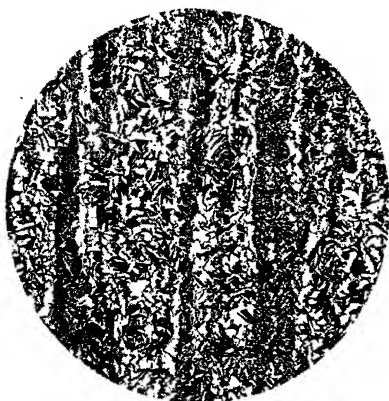
FIG. 29.—STEEL CONTAINING 0.12 PER CENT. C, 1.46 PER CENT. MN, HOT ROLLED ANNEALED AT 1750° F. FOR 1 HR. $\times 150$. ETCHED WITH HNO_3 . (C. H. CHOU.)

FIG. 30.—STEEL CONTAINING 0.12 PER CENT. C, 2.58 PER CENT. MN, HOT ROLLED, ANNEALED AT 1750° F. FOR 1 HR. $\times 150$. ETCHED WITH HNO_3 . (C. H. CHOU.)

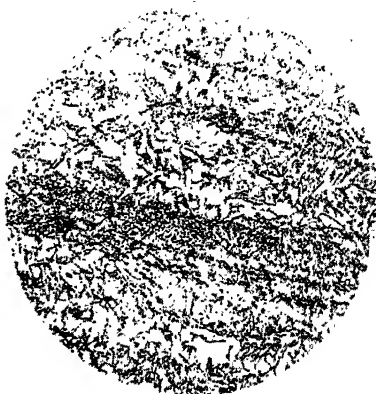
FIG. 31.—STEEL CONTAINING 0.31 PER CENT. C, 2.65 PER CENT. MN, HOT ROLLED, ANNEALED AT 1750° F. FOR 1 HR. $\times 150$. ETCHED WITH HNO_3 . (C. H. CHOU.)

Reduced to approximately $\frac{2}{3}$ original size.

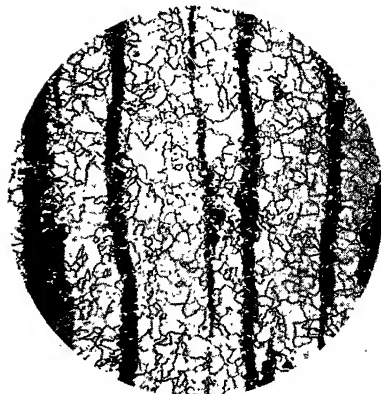
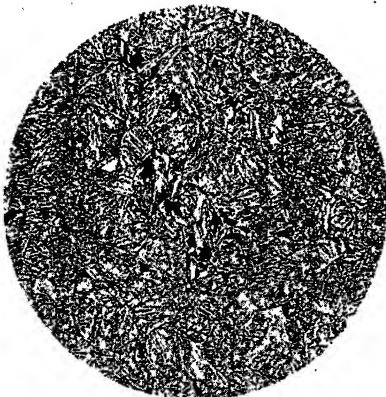
26



27



28



30



31



Annealing at 1650° was effective in entirely breaking up the cast structure of a sample of 0.12 per cent. carbon and 1.46 per cent. manganese and even as high in manganese as 2.0 per cent. With steels of higher manganese content it was found that a temperature of 1850° F. maintained for one hour per inch was necessary for the same purpose, since 1650° F. had very little effect on steel. With 0.31 per cent. carbon and 2.65 per cent. manganese, however, longer time of annealing was imperative; the structure obtained after one hour at 1850° F. (Fig. 23) being not quite uniform. After much further study, it was found that as a general rule the question of breaking up the "cast structure" is merely a matter of time and temperature.

A few unusual features, however, were sometimes observed. A piece of steel of about 0.10 per cent. carbon and 1.5 per cent. manganese, "as cast," was heated at 1850° for one hour and cooled in the furnace. The structure of the sample, after usual preparation, was found to consist of a pronounced network made up of sorbite while the space between the network was occupied by ferrite (Fig. 24). The heat treatment was repeated, the sample held at 1850° for 2 hr. and cooled in the furnace. It goes without saying that the decarburized layer was removed, and yet the structure remained almost the same (Fig. 25). The writers do not desire to offer any speculative explanation of the observed phenomena, but as the heating for 2 hr. at 1850° should have been sufficient to produce uniform structure (because the heat-treated samples were only $\frac{1}{2}$ in. thick) a probable conclusion seems to be self-suggestive; namely, that this steel must be fairly slow in its process of attaining perfect homogeneity.

There is yet much to be learned in regard to the proper making, pouring, and treatment of low-carbon manganese steels, and further investigation will undoubtedly permit us to understand better the nature and the cause for the unusual structures observed.

Banded Structure in Hot-rolled Steels

The samples from the hot-rolled bars of steels previously studied in the "as-cast" state were next examined. Fig. 26 shows the structure of 0.12 per cent. carbon and 1.46 per cent. manganese; Fig. 27 that of 0.12 per cent. carbon and 2.58 per cent. manganese, and Fig. 28 that of 0.31 per cent. carbon and 2.65 per cent. manganese. The banded structure is much more pronounced in the first two steels. The bands consist of small cementitic particles, embedded in ferrite. The space between bands is occupied by ferritic and sorbitic grains.

Holding 1-in. round bars of these steels at 1650° F. for one hour, as well as heating to 1750° F. or even 1900° F., followed by furnace cooling, does not eliminate the banded structure; on the contrary, the bands become more pronounced. For an illustration, the structures of steels after heating to 1750° F. will be found in Figs. 29, 30 and 31. It looks as

though the grains containing carbon have agglomerated and left the space between bands nearly free of carbon. Consequently, the structure consists of a ferritic network with sorbitic bands running through it.

C. H. Chou, a graduate student at Carnegie Institute of Technology, made a comprehensive study of the banded structure in low-carbon manganese steels, but his interesting findings can be mentioned only briefly. It was established that steels containing approximately 0.15 to 0.25 per cent. carbon and 1.0 to 2.25 per cent. manganese all show banded structure on annealing after hot rolling at 1600°, 1700°, and even 2000° F. (see Fig. 32). On normalizing these steels at the same temperatures, the banded structure disappears (Fig. 33). On reheating followed by

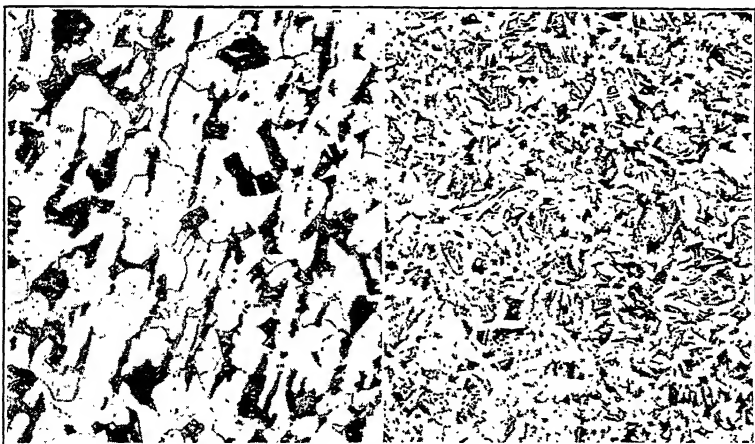


FIG. 32.—STEEL CONTAINING 0.20 PER CENT. C, 1.03 PER CENT. MN, HOT ROLLED, ANNEALED AT 1600° F. FOR 1 HR. $\times 100$. ETCHED WITH HNO_3 .

FIG. 33.—STEEL CONTAINING 0.20 PER CENT. C, 1.03 PER CENT. MN, HOT ROLLED, NORMALIZED. $\times 150$. ETCHED WITH HNO_3 . (C. H. CHOU.)

furnace cooling, the bands are again in evidence. The only visible difference is that the size of the ferritic grains increases with the higher temperature and longer period of treatment.

The occurrence of banded structure, or "ghost lines," has been generally linked with phosphorus segregation. Consequently every known method has been employed to ascertain whether phosphorus could be responsible for banded structure. Chemical analysis, preferential etching, etc., failed to produce any evidence of existent phosphorus segregation.

Somewhat similar results were obtained in recent work in England.²¹ The steels included in that report are of similar chemical composition

²¹ R. D. Report No. 61, Research department, Woolwich Arsenal, England.

and showed no banding either on quenching or normalizing. On annealing, however, some of the steels showed bands, depending on the rate of cooling; if cooled after annealing at the rate of 10° per min., the bands are almost invisible, but at slower rates (6.2, 5.3 and 0.75 per min.) they are quite evident.

It should be mentioned that even when the annealing of cast steels failed to produce a homogeneous and uniform structure, the mechanical properties of these steels were found unimpaired. Similarly, the banding as displayed by hot-rolled steels has apparently no detrimental effects on their properties.

The question of banding, often confused with "ghost lines," has lately received a great deal of attention. The views of many noted British metallurgists given in a recent symposium²² are somewhat diversified, but the consensus of opinion is that banding is not as detrimental to the quality of steel as was formerly believed.

SUMMARY

1. Low-carbon manganese steels can be manufactured in either basic or acid open-hearth or electric furnaces, available manganese alloys giving any desired composition.

2. The finishing of a manganese heat in the furnace, and the use of silico-manganese for making low-carbon heats involve special problems in furnace operation which are not yet entirely solved and deserve further study.

3. The lower critical point A_1 (carbon point) on heating 1350° – 1360° F. is independent of both carbon and manganese content within the limits of 0.07–0.31 per cent. carbon and 1.50–3.15 per cent. manganese.

4. The two upper critical points, designated as A_{2-3} , in all probability combined, on heating occur at almost the same temperature, 1480° – 1500° F.

5. The A_1 and A_{2-3} points on cooling are lowered as percentage of manganese increases, but remain approximately the same for the steels containing similar amounts of manganese, independently of carbon content.

6. The sluggishness of transformations is very apparent; in some cases the steels cool through a range of 100 degrees or more before the transformation is complete.

7. It is to be expected that steels of these types, because of slow, sluggish transformation, should be well suited for a large variety of heat treatments.

8. Low-carbon manganese steel may be useful chiefly in the field of the cheaper alloy steels, where large tonnages are desired of a steel with properties superior to those of ordinary open-hearth carbon steels.

²² *The Metallurgist* (Supplement to *The Engineer*, Aug. 27, 1926).

9. A double heat treatment is necessary to bring out the best properties in these steels.

10. If properly heat treated, low-carbon manganese steels will give values for strength and toughness comparable to those in the lower nickel and nickel-chromium steels. They are characterized by an indefinite yield point and are markedly insensitive to abuse in heat treatment, especially in regard to ductility.

11. There are indications of superior fatigue and shock resistance, also of superior physical properties at high temperatures in these steels, but the collection of data on this subject has only just begun and no definite statement can be warranted.

12. The writers are inclined to the belief that both the sorbitic network (as described in annealed cast steels) and the banding (in hot-rolled samples) are closely related to the mechanism of solidification. Also, certain treatments, or more precisely, the manner of cooling, are more prone to develop banding, which is an indication of the complexity of the reaction going on in steels during cooling from high temperatures. It is felt, however, that a careful study of the questions of equilibrium in the solid state at various temperatures and the interactions between different elements must be made before a sound theory can be advanced.

ACKNOWLEDGMENTS

The writers wish to thank C. E. Sims, formerly of the Bureau of Mines, who supervised and directed the making of steels. Because of the pressure of work in a new position he was unable to participate directly in the preparation of this paper, but many of the data summarized are the result of his work on the problem.

The writers thank C. H. Chou for permission to use some of his photomicrographs.

Desulfurizing Action of Manganese in Iron

By C. H. HERTY, JR.* AND J. M. GAINES, JR.,† LACKAWANNA, N. Y.

(Pittsburgh Meeting, October, 1926)

INTRODUCTION

It is a matter of frequent record in the literature that manganese will desulfurize molten iron under proper conditions. Particularly has much been written with regard to this action in the mixer, and numerous reports have been published describing experiments in which advantage was taken of this phenomenon in order to obtain a purer iron for the open hearth.

The question of sulfur elimination in the ladle has, however, received comparatively little attention. The reaction by which sulfur is removed is considered to be:



The manganese sulfide, being but sparingly soluble in the iron, and of a lower density, will rise from the body of the metal and collect as a slag on the surface. It is therefore an action taking place solely in the metal, and independent of interaction between slag and metal. As will be shown later the presence of slag on the metal tends to offset the effect of manganese elimination. It is this fact which distinguishes desulfurization in the ladle from desulfurization in the mixer, for it has been shown¹ that the amount of sulfur eliminated in the mixer is closely tied up with the composition of the slag.

EXPERIMENTAL PROCEDURE

In order to ascertain the extent of and the factors controlling this elimination in the ladle, several casts were sampled carefully at the blast furnace and again when the ladles were being emptied, either at the mixer, open hearth, or foundry. If the sulfur content varied from ladle to ladle at the blast furnace the average sulfur content of each ladle was obtained by integrating the curve "sulfur vs. time" for the cast under

* Research associate, Massachusetts Institute of Technology, School of Chemical Engineering Practice, Bethlehem Steel Co.

† Assistant director, Massachusetts Institute of Technology, School of Chemical Engineering Practice, Bethlehem Steel Co.

¹ S. V. Williams: Control of Sulphur in the Basic Open-hearth Process, *Blast Furnace and Steel Plant* (Jan., 1923) 11, 51.

consideration. The samples at the terminal point were taken from as close to the middle of the ladle as possible.

EXTENT OF ELIMINATION OF SULFUR

The results show that within the limits of error in sampling and analysis, if any elimination takes place, the final content of sulfur and manganese (*i. e.*, the amount in the metal when the ladle reaches the mixer, etc.) will be related as follows:

$$(\text{Per cent. manganese}) (\text{per cent. sulfur}) = 0.070,$$

provided the product (per cent. Mn) (per cent. S) is greater than 0.070 at the blast furnace. In other words, the higher the manganese, the lower

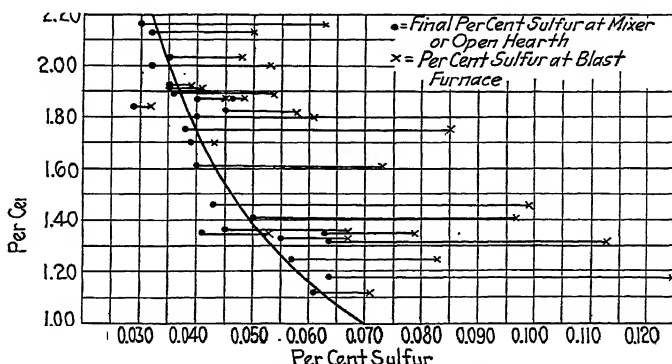
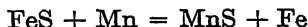


FIG. 1.—DESULFURIZING ACTION OF MANGANESE IN PIG IRON.

the sulfur will be after the elimination has ceased. Fig. 1 has been included to show graphically this relationship. It also shows just how much sulfur was eliminated from each ladle tested.

If there is no source of sulfur, such as blast furnace slag, in the ladle, it is evident, theoretically, that when the product (per cent. Mn) (per cent. S) is below 0.070 at the blast furnace, no elimination of sulfur should take place. Under these conditions the reaction



should reverse and if sulfur were present in a second phase, resulfurization of the iron would occur. However, when no external source of sulfur is present the sulfur content of the iron will remain constant.

EFFECT OF TEMPERATURE AND TIME

The effect of temperature on the relationship (per cent. Mn) (per cent. S) = 0.070 could not be definitely determined, as the temperature at the terminal point varied but little from 2400° F., the average temperature

for the casts. However, it may be stated, qualitatively, that since the reaction



is exothermic, *i. e.*, gives off heat, the lower the temperature the further will the reaction go to the right with formation of MnS. The practical limits in temperature, as far as desulfurization is concerned, will be the point at which the iron becomes so viscous that the MnS can not rise to the surface.

As to the influence which the time interval between casting and pouring has upon the amount of desulfurization, it may be said that after 1 hr. elimination has practically ceased. All of the ladles from which samples were taken had stood for at least this length of time, and some for as long as 2 or 3 hr.

PERCENTAGE ELIMINATION OF SULFUR

The percentage elimination of sulfur between the blast furnace and the mixer depends on the sulfur and manganese content of the iron at the blast furnace and on the manganese content at the mixer. For example, take a manganese content of 2 per cent. at the blast furnace and assume that this has dropped to 1.95 per cent. at the mixer. Since the product (per cent. Mn) (per cent. S) = 0.070 at the mixer the amount of sulfur at the mixer will be $\frac{0.070}{1.95} = 0.036$ per cent. S. Then with varying sulfur contents at the blast furnace the percentage elimination of sulfur will be:

Manganese at Blast Furnace, Per Cent.	Sulfur at Blast Furnace, Per Cent.	Elimination of Sulfur, Per Cent.
2.00	0.070	48.5
2.00	.060	40.0
2.00	.050	28.0
2.00	.040	10.0
2.00	.036	0

Similarly for a manganese content of 1.50 per cent. at the blast furnace which we will assume gives 1.45 per cent. Mn at the mixer, the sulfur content of the iron at the mixer will be $\frac{0.070}{1.45} = 0.048$ per cent. S. The percentage elimination of sulfur will in this case be:

Manganese at Blast Furnace, Per Cent.	Sulfur at Blast Furnace, Per Cent.	Elimination of Sulfur, Per Cent.
1.50	0.080	40.0
1.50	.070	31.4
1.50	.060	20.0
1.50	.050	4.0
1.50	.048	0

The percentage elimination of sulfur depends not only on the manganese content of the iron but also on the sulfur content, the higher the product (per cent. Mn) (per cent. S) at the blast furnace, the greater the percentage elimination of sulfur for a given manganese content of the iron. It can be very easily shown that a 1 per cent. Mn iron may show a greater percentage elimination of sulfur than a 2 per cent. Mn iron provided the sulfur content of the first is sufficiently high. The final sulfur content of the high-manganese iron will, however, be the lower.

Manganese, Per Cent.	Initial Sulfur, Per Cent.	Final or Equilibrium Sulfur, Per Cent.	Sulfur Eliminated, Per Cent.
2.00	0.040	0.035	12.5
1.00	.120	.070	41.5

For a given initial sulfur, the higher the manganese the greater will be the percentage elimination of sulfur, provided the product (per cent. Mn) (per cent. S) is greater than 0.070 at the blast furnace in each case.

EFFECT OF BLAST FURNACE SLAG IN THE LADLE

During the first few runs, the procedure consisted in sampling each ladle as the metal was cast at the furnace. This was later modified to include only the first two ladles, because it was found that after the slag had begun to run from the furnace—usually during the filling of the third ladle—a certain amount of it would invariably be drawn under the skimmer and collect on the surface of the metal in the ladle. Since this condition was especially undesirable, it was deemed necessary to restrict sampling to the first two ladles.

The effect of slag on the metal in the ladle is well illustrated by the data given below. These represent an attempt to determine the drop in the sulfur in the third and fourth ladles, when it was observed that there was a thin stream of slag on the metal after passing the skimmer.

Cast No.	Ladle No.	Sulfur		Manganese		Increase in Sulfur Per Cent.
		Initial Per Cent.	Final Per Cent.	Initial Per Cent.	Final Per Cent.	
17	1	0.043	0.039	1.59	1.70	— .004
	2	.063	.059	1.54	1.70	— .004
	3	.063	.088	1.51	1.57	+ .025
	4	.059	.072	1.48	1.46	+ .013

LOW-MANGANESE IRON

A cast of foundry iron, with very little manganese in the metal, was sampled at blast furnace and foundry. Here the product (per cent. Mn) (per cent. S) at the blast furnace was 0.025 and 0.023 for the first and second

ladles of the cast and the samples secured at the foundry showed no elimination of sulfur. The results were:

Cast No.	Ladle No.	Sulfur		Manganese	
		Initial Per Cent.	Final Per Cent.	Initial Per Cent.	Final Per Cent.
20	1	0.055	0.062	0.33	0.40
	2	.066	.069	.22	.34

It will be noticed that there was an increase in manganese and sulfur in the metal. This may be due to discrepancies in sampling or to the presence of a slight amount of furnace slag on the metal, although these samples are for the first two ladles only.

APPLICATIONS

Of practical importance is the question of skimming the slag from the metal, not only at the furnaces, but also at the time of pouring the hot metal ladle. It can readily be seen that the advantage derived from the elimination of sulfur as manganese sulfide will be partially lost if the sulfide is poured into the open hearth, the sulfur entering the slag rather than the metal. The highly oxidized slag which is carried in the furnace will cause most of the sulfur in the slag to be returned to the metal.

With respect to sulfur the importance of high-manganese pig iron to both the blast furnace and open-hearth operator can readily be seen. High-sulfur, low-silicon iron is economical in both the blast furnace and the open hearth, provided the sulfur may be eliminated to the desired point in the final steel. High-manganese iron serves to bring about this elimination before the iron is charged into the open hearth.

CONCLUSIONS

Within the range covered by these data, the following conclusions are drawn:

1. In ladles of molten pig iron the equilibrium between manganese and sulfur at 2400° F. is expressed by the relation

$$(\text{per cent. Mn}) (\text{per cent. S}) = 0.070,$$

when the product (per cent. Mn) (per cent. S) is above 0.070 at the blast furnace, and when no blast furnace slag is present on the iron.

2. If the product (per cent. Mn) (per cent. S) is above 0.070 at the blast furnace elimination of sulfur will take place until equilibrium is established.

3. If the product (per cent. Mn) (per cent. S) is less than 0.070 at the blast furnace little or no elimination of sulfur will take place.

4. If blast-furnace slag is present in the ladle sulfur may be reduced from the slag into the metal.

5. If the manganese sulfide eliminated from the iron is poured into the open hearth the advantage of desulfurization by high manganese is lost.

ACKNOWLEDGMENT

The authors wish to express their appreciation for assistance rendered by the blast-furnace department of the Lackawanna plant of the Bethlehem Steel Co.

DATA FOR DESULFURIZATION

The word "initial" signifies the sample taken at the cast. 'Final' signifies the sample at open hearth or mixer.

Cast No.	Ladle No.	Manganese		Sulfur		Temperature °F.		Minutes in Ladle
		Initial	Final	Initial	Final	Initial	Final	
3		2.16		0.063	0.030			
9		2.35	2.13	.050	.032			
10		2.22	2.03	.048	.035			
16	2	1.83	1.92	.039	.035	2695	2420	97
6		2.10	2.00	.053	.032			
20	2	1.90	1.92	.041	.035	2645	2335	246
11		2.38	1.89	.054	.036			
20	1	2.00	1.87	.045	.040		2445	
5		1.87		.048	.046			
16	1	1.86	1.84	.032	.029	2650	2445	107
8		2.05	1.82	.058	.045			
7		1.90	1.80	.061	.040			
4		1.75		.085	.038			
17	1	1.59	1.70	.043	.039	2630	2345	196
12		1.89	1.61	.073	.040			
2		1.46		.099	.043			
19	2	1.69	1.41	.097	.050	2595	2445	66
13		1.41	1.36	.067	.045			
14		1.41	1.35	.053	.041			
1	2	1.34	1.35	.079	.063	2610	2425	140
1	1	1.32	1.33	.067	.055	2560	2425	153
15	2	1.31	1.32	.113	.064			91
18	2	1.49	1.25	.083	.057	2535	2420	162
15	1	1.31	1.18	.125	.064			82
18	1	1.26	1.12	.071	.061	2565	2395	182

DISCUSSION

K. C. McCUTCHEON, Ashland, Ky.—Mr. Gaines, is this manganese sulfide visible on top of the ladle?

J. M. GAINES.—You can see some slag. You know a ladle is generally well covered with coke and it is hard to distinguish the slag.

K. C. McCUTCHEON.—When you refer to slag on the iron, do you mean blast-furnace slag? Is there any other slag that you think would affect it?

J. M. GAINES.—Any slag that contains sulfur would affect the sulfur content of the metal.

Iron-Manganese Alloys Low in Carbon*

BY SIR ROBERT A. HADFIELD, Bt., D. Sc., D. Met., F. R. S., F. I. C., M. I. C. E.†
(Cleveland Meeting, April, 1927)

Table 1 gives the composition of the specimens listed by the author. The details of test procedure and the results of the tests on mechanical,

TABLE 1.—Composition of Test Specimens.

Specimen No.	Specimen Mark.	Analysis, per cent.					Iron (by diff.)
		Carbon.	Mn.	Phosph.	Manganese.	Manganese.	
1	1277D	0.11	0.30	0.046	0.069	0.06	99.82
1	1279A	0.07	0.03	0.190	0.070	1.88	98.03
2	1279B	0.06	0.08	0.096	0.088	3.96	96.73
2	1279B ¹⁴	0.10	0.16	0.063	0.066	4.10	96.53
2	1279B ¹³	0.11	0.19	0.047	0.049	4.82	94.78
2	1279C ¹¹	0.06	0.12	0.061	0.064	6.08	92.90
2	1279C ¹²	0.06	0.06	0.076	0.064	6.76	92.96
2	1279D	0.07	0.10	0.064	0.069	9.45	90.24
3	1278E	0.06	0.12	0.103	0.060	13.96	96.60
3	1278E ¹³	0.14	0.26	0.061	0.064	14.30	88.21
4	1279G	0.12	0.35	0.060	0.061	17.10	83.29
4	1278F	0.06	0.18	0.054	0.048	22.70	76.99
4	1278H	0.20	0.70	0.065	0.038	33.90	60.11
5	1278I	0.29	1.36	0.064	0.060	83.50	14.76

magnetic and electrical properties and corrodibility of the alloys are published in full in a paper by the same author, "Alloys of Iron and Manganese Containing Low Carbon," presented before the Iron and Steel Institute (of Great Britain) and published in full in their *Proceedings* for 1927, to which reference should be made. After reporting these tests the author gives the following discussion of results:

The studies here reported do not indicate that any of the series has really valuable or useful qualities for industrial applications, though the demands of industry are so varied that it is unsafe to prophesy that no special use will be found for a given material. The non-magnetic property of manganese steel is due to the association of Fe and Mn, but carbon has also an effect, enabling non-mag-

netic qualities to be obtained with a much lower percentage of Mn than would otherwise be the case; for example, 16 per cent. Mn is necessary for alloys practically free from carbon, while with 1 per cent. C non-magnetic alloys may be obtained with as little as 7 per cent. Mn. Carbon, however, is the prime cause of the development of magnetic qualities by heat treatment. (See section 8 of the paper referred to above.) These Fe-Mn alloys show a resistance up to 87 microhms per c. c., but as there are other alloys which have a resistance as high as 130 microhms per c. c., this does not provide any new feature of value.

The most marked feature of the series studied is the brittleness in the range 3 to 10 per cent., with a maximum of brittleness at 7 per cent. Material with this percentage, while possessing quite appreciable hard-

* Summary of a manuscript presented to the Institute and discussed at the Cleveland Meeting.

† Honorary Member, A. I. M. E.

ness and cohesion, is incapable of much deformation, and, further, the nature of the cohesion is such that it is apparently easily influenced by small factors. What these factors are is not at present known. On either side of 7 per cent. the alloys partake of this character, more or less according to their proximity in composition thereto, but finally merging into the characteristics of high tenacity and ductility previously referred to at the higher percentage, and, on the other hand, towards the low tenacity and comparatively high ductility of iron itself at the lower percentages.

In the author's original research on alloys of iron, manganese and carbon, in which the carbon percentage in relation to that of manganese was approximately as 1 to 10, a somewhat similar characteristic, that is, of hardness and brittleness, was specially noted, i. e., at what was then known as the brittle percentage containing about 5 per cent. manganese. This particular alloy containing about 0.5 per cent. carbon was so brittle that it could easily be pounded with a hand-hammer, its hardness being over 600 Brinell, sufficient to scratch glass. Here again the brittleness of this alloy appears to be connected with a similar brittleness found, although in not so marked a degree, in the alloys of iron and manganese with low carbon content containing from 4.00 to 10.00 per cent. manganese. While the presence of carbon undoubtedly adds to the hardness and brittleness in such alloys of comparatively low manganese percentage, this characteristic is evidently derived mainly from the combination of the iron and manganese.

Quenched manganese-iron alloys containing low carbon behave altogether differently as compared with the iron-manganese-carbon alloy known as manganese steel. In these low-carbon alloys little or no improvement is obtained by quenching. This is illustrated by specimen 1379 E, which contains only 0.09 per cent. of carbon along with 12.95 per cent. manganese, and in which the elongation produced by tensile stress is under 2 per cent., as compared with no less than 70 per cent. for manganese steel (1.25 per cent. carbon and 13 per cent. manganese). Thus these results further emphasize the exceedingly important part played by carbon. They are almost as startling as the well-known fact that carbon steel containing 0.70 to 1.00 per cent. carbon when quenched in water from a temperature of about 750° to 800 °C. becomes exceedingly hard and possesses 600 to 800 Brinell hardness number, whereas carbon steel containing only 0.09 per cent. carbon; that is, a similar amount to that in the manganese-iron alloy represented by the specimen 1379 E just described, even if heated to 1100 °C. and quenched in cold water, remains unhardened. In manganese steel, which contains a high percentage of carbon, extraordinary toughness results from quenching, whereas in carbon steel hardness and brittleness result; or in the manganese-iron alloys with low carbon and up to about 30 per cent. manganese,

quenching from high temperatures not only produces no increase in hardness, but leaves the material untoughened and in a comparatively brittle condition.

DISCUSSION

J. H. HALL, High Bridge, N. J.—Sir Robert Hadfield's researches have been of the utmost value, as we all know, and he continues to inform us about the properties of the steel which he originally developed—Hadfield's manganese steel—containing $1\frac{1}{4}$ per cent. carbon and high manganese. He has been indefatigable in ascertaining more facts about it, chiefly scientific, but many of them, as I can testify as a manufacturer of that steel, of great commercial value. We manganese steel makers in this country are glad to see Sir Robert interesting himself in the possibility of steels containing manganese, but with much different composition from the steel which he originally developed. That steel had one valuable property which was the ability to resist most kinds of wear. It is that property almost entirely which has made it useful.

A great deal of work has been done in this country on steels containing varying percentages of manganese, with carbon ranging from 0.20 to 0.40 per cent. There is no thought in the minds of metallurgists that they will duplicate, or even resemble, Hadfield's 12 per cent. manganese steel. It is the thought rather to develop these lower manganese steels as substitutes for nickel and nickel-chrome steels of medium carbon content. It is not to be expected that these steels will possess the property of resisting wear, such as 12 per cent. manganese steel stands up to so successfully.

These steels, containing from 0.20 to 0.40 per cent. carbon, and from 1.25 to 2.5 or even 3 per cent. manganese, have remarkable physical properties which make them of great and increasing commercial importance. It is to be regretted, therefore, that Sir Robert has confined his investigation to steels containing such a very low percentage of carbon, and it is to be hoped that he will undertake a similar investigation of steels with a somewhat higher carbon content, such as are already being manufactured in quantity in this country at the present time.

R. A. HADFIELD (reply to discussion).—I note with interest Mr. Hall's remarks on my paper, and as he mentions that much work has been done in America with regard to iron-manganese alloys containing about 0.20 to 0.40 per cent. carbon, I would suggest that he add particulars of the results obtained, including analyses and mechanical tests. We have worked on quite a large scale with steels containing 1.50 to 2 per cent. manganese, and with proper heat-treatment they give very remarkable results.

Upon reference to my experimental record book, I find that we obtained the following results from my experimental steel No. 1772 (Dec. 12, 1902), that is from a forged bar. This steel contained 0.53 per cent. carbon and 1.56 per cent. manganese. By heat-treatment I was able to get in one case 38 tons per sq. in. elastic limit; 52 tons maximum breaking load; 23 per cent. elongation; reduction of area, 57 per cent; Brinell ball hardness number, 248, also other similar results.

If Mr. Hall has the results of the tests on steel containing about 2.50 to 3.00 per cent. manganese, to which he refers, it would be interesting if he would also contribute these figures to the discussion on this paper. There seems to be a certain limit to such material, that is the carbon must not be too high. However, he may have results showing otherwise.

As regards continuing my experiments on some further new sets of this type of steel, but containing higher carbon, such researches are costly and I am inclined to think, from tests of my own made a long time back, that the results would not be particularly favorable.

Need for Research in Foundry Pig Iron*

BY RICHARD MOLDENKE,† WATCHUNG, N. J.

(New York Meeting, February, 1927)

SO FAR as the quality of the product is concerned, the history of the production of pig iron for foundry purposes is one of constant retrogression. The steps in this deterioration began with cold-blast charcoal pig iron, then anthracite iron, coke iron; then gradually warm to hot-blast coke iron, with the charcoal furnaces also heating their blast to get greater tonnages, and finally the present-day hot-blast coke irons with scrap additions to the ordinary burden that, according to one European report, have gone as high as 65 per cent.

Parallel with this quality retrogression is the enormous increase in furnace tonnages from about 15 to over 1000 tons per day in exceptional instances of modern practice. The key to this situation is economic pressure. The effect is a growing differentiation between furnace production for gray iron and malleable foundries and for the production of commercial steel. As the foundry is often compelled to draw upon the pig irons made for steel purposes, for economic reasons, whereas the steel industry does not draw upon foundry pig irons proper, the situation is resolving itself into one of grave consequence for the life of the gray-iron castings industry, and hence the intensive present-day feeling in foundry circles for rigid specifications to govern the quality of pig irons sold for foundry use.

In the discussion of a paper on the use of scrap in the blast furnace held recently in the Institute, a very pertinent remark was made by one of the participating furnacemen, who confessed that "the things found wrong by the foundryman with his iron were beyond his depth." It may be added that those things are also beyond the depth of most foundrymen who, however, know from their results that they are getting poorer castings today with many pig irons than they did with those same brands in former years. Under the former fracture buying, the foundryman at least could select the strongest metal, as judged by the sledge, from various brands of similar fracture and pig section, and his results proved

* This paper and the three following papers, by W. B. Coleman, Enrique Touceda and Ralph H. Sweetser, respectively, were presented at a Round Table on Carbon in Pig Iron, held at A. I. M. E. headquarters, Feb. 16, 1927, under the chairmanship of Mr. Sweetser. Although scheduled for only one session it was necessary to hold a second session in order to complete the discussion. For the discussion at the second session see p. 482.

† Consulting metallurgist.

the soundness of his conclusions. With the advent of iron analysis, and in spite of the growing uniformity of the metal through greater tonnage production, furnace attention was diverted from quality production to obtaining greater yields of iron within a given analysis range.

How the foundryman had to meet this situation, apart from setting his own house in order on the score of production practices, is best illustrated by the action of one very prominent foundryman who took the entire output of two Southern cold-blast charcoal furnaces as long as they ran, no matter what the analyses were. By admixing a small percentage of this very high-class pig iron, which did not in the least affect the analyses of the rolls he made, he obtained for his product a much longer wear in service than others obtained who did not so improve their mixtures. The reason for this situation is what foundries and furnaces would like to know, and is the best argument for a concerted drive for *research* on the part of both industries working together to a common end.

CONDITIONS UNDER WHICH PIG IRON IS MELTED

Perhaps an analysis of the operating conditions under which pig irons are remelted will give a clearer insight into the bearings of the problem. The processes carried out in the blast furnace are supposed to be of a strictly reducing character. Ores are deprived of their oxygen to yield metallic iron and what manganese may be present. Similarly silica from fuel ash and ore gangue yields silicon in proportion to the temperatures existing, while carbon is taken up from the fuel by the melting iron sponge. Sulfur and phosphorus are accompanying unavoidable ingredients. If reduction has been complete, there should be no evidences of residual oxides, or even a reoxidation at the tuyeres. This is in contradistinction to the effect of all remelting processes, which are more or less oxidizing and, as used in foundries in the production of castings, yield a product that exhibits somewhat different physical characteristics than metal poured into molds "direct" from the blast furnace, even with the same composition in both instances. This has led to the term "virgin metal" given to pig iron from the blast furnace, whereas the remelted pig iron going into castings, sprues, etc., and finally ending in the scrap pile is called "remelted stock." The proportioning of pig to scrap in foundry mixtures is one of the important elements in the production of castings with desired qualities, the same analysis with higher pig percentages giving softer castings as a general rule. The use of scrap to excess in the furnace delivers metal having some of the characteristics of remelted stock to the foundry, and hence upsets the calculations of the foundryman in his own work.

When pig iron is used for steel making, whether by the Bessemer, open-hearth or electric-furnace process, a deliberate oxidation of the

metal is effected to remove the carbon, etc. This leaves the metal so badly oxidized, however, that it must be deoxidized with silicon or manganese, and thus wind up with a reducing process. This is possible because of the high temperatures prevailing that are not attained in the ordinary foundry melting processes. It may, therefore, not be essential for furnaces supplying the steel industry to aim for the high quality necessary for foundry purposes. But for the gray iron and malleable foundries, in which the molten metal cannot be given a deoxidizing finish with any degree of certainty, the pig irons used must not have been subjected to any oxidizing influences in the making at the furnaces, for they get enough oxidation in the foundry melting operation later on. Although it should be the business of the foundryman to conduct his melting operation so that the very smallest degree of oxidation takes place, for melting is all that he wants, it will be readily understood that the better the material he starts with, his own operations being normal and under the best conditions possible, the better will be his product.

Until the time comes, therefore, when it is possible successfully to deoxidize molten foundry iron, pig irons must be able to stand enough of the unavoidable oxidation incident to normal foundry practice, to allow the making of satisfactory castings, if the foundry is to live. That many of the foundry pig irons of today will not fill this requirement is patent, and inquiry need only be made of manufacturing concerns processing the castings they purchase or make, to learn that without change in practice, results have become more irregular than ever before and can only be corrected by cutting out offending pig irons. Where castings do not enter into conditions of actual service this may not be so apparent, and may account for good reports received by furnaces on their high-scrap pig irons. But the acid test of machining the castings and putting them to work quickly brings out the facts.

The ordinary analysis of cast iron does not tell the whole story by any means. The foundryman has no way of telling whether the furnace from which he may get some of his iron is running normal or working badly. The analysis will not tell him this unless it is complete enough and the separate determinations have been carried through accurately. Thus, high-sulfur and low-total carbon will ordinarily show up an "off" iron. Furnaces, however, avoid carbon determinations if possible, and with the volumetric sulfur method, and the possible presence of such elements as copper, which influence this method seriously, the foundryman usually finds himself in trouble without having known beforehand that this or that pig iron could bear watching closely.

The time may come when a reported analysis will include oxygen of dissolved iron oxide and of occluded gases separately, nitrogen, possibly hydrogen, surely chromium, copper, nickel and many other elements present in small but appreciable quantities. The information gained

from the study of catalytic action by small percentages of foreign material in a substance will eventually throw light upon the mystery of "iron without life" in the foundry. Why is it that oxidized iron freezes so quickly in spite of a high degree of superheat that may be available? There is much yet to learn on the question of a real analysis of pig and cast iron, and better knowledge on the subject will enable furnace and foundry to search for ways of avoiding troubles, or where not possible to do this, to correct them.

The trend today is toward a specification of the total carbon content in pig irons. Formerly this was the exception and fixed only when it was essential that the irons be of the highest character for very special purposes. It is well known that the normal total carbon content of any pig iron depends on the silicon content, the temperature and operation of the furnace, in lesser measure the content of the other elements, and doubtless other matters of furnace practice better known to the furnace operator. Would it not be wise, then, for the furnaces to study this question and settle, together with the foundrymen, a minimum for total carbon for given silicon ranges, so that there may be no eventual necessity of combatting freak specifications of this rather difficultly controlled element later?

There remains a final consideration. Foundrymen should be willing to pay for well-made pig irons. As a matter of fact, it will be found that wherever it is essential that the so-called "honest" pig irons, that is, pig irons made with more than enough coke, be used, there will be found a willingness to pay a reasonable premium for them. That this is the case may be seen from the continued production of charcoal pig irons, of the extending use of the special nickel-chrom (Mayari) pig irons. All these irons sold above the market price of the regular run of foundry pig irons. Indeed it is to be hoped that furnaces will cater specially to discriminating foundrymen, and give them lines of pig iron specially suited to their requirements, such as irons for making piston rings, Diesel engine cylinders and heads, or high-test cast iron for various purposes.

The recent developments in Germany in superior quality castings, particularly if the results obtained there can be duplicated with our pig irons, will call for extra quality in the pig irons and, strangely, that they be not only machine-cast but that they be chilled by water in addition. This may seem iconoclastic to the old-time foundryman, but looks very much as if the machine casting of pig iron has not yet gone far enough.

Should the proposal to have furnaces and foundries cooperate in a joint study of the testing of and specification for foundry pig iron be taken up seriously, much good will result to these branches of the great iron industry. Although the foundrymen have their excellent organization and can readily undertake lines of tests cooperatively, study of furnace operation seems to have been confined to individual investigators con-

nected with industrial entities not closely related. Much has been published here and abroad, but it would seem to have been overlooked or disregarded, such as investigations by very capable talent on the reoxidation of iron in the blast furnace as it passes the tuyere region. And yet this may have a vital bearing on the desired quality of foundry pig irons. It would, therefore, seem time for the technical side of the furnace development of the country to be organized into associated effort, so that this industry may benefit in similar manner as has that of the foundry. With these two interests doing effective research work, cooperation will inevitably result.

DISCUSSION

R. FRANCHOT, Washington, D. C.—Dr. Moldenke speaks of “honest” pig iron as one that is “made with more than enough coke,” and then praises charcoal pig iron. Is not that inconsistent? Charcoal practice uses very much less carbon per unit of iron than coke practice, as a rule, and it would seem that perhaps the virtue of charcoal iron is due to less carbon rather than more.

R. MOLDENKE.—The amount of carbon you put into a blast furnace when using charcoal is less than when using coke. To melt iron in the cupola requires almost four times as much charcoal as it does coke.

T. L. JOSEPH, Minneapolis, Minn.—Mr. Franchot is right. Charcoal is usually measured in bushels and the charcoal would probably run from 1700 to 2000 lb., per ton of iron. The fixed carbon in charcoal is less than the fixed carbon in coke, largely because of volatile matter in charcoal. Therefore less carbon is required in charcoal practice than in coke practice.

R. MOLDENKE.—Most of the charcoal iron we use is low-silicon (the silicon is down to 0.10) and does not require as much fuel as the high-silicon iron. Is it not a fact that a smaller amount of charcoal is used because you do not need so much heat in the furnace? A 2.80 or 3 per cent. total silicon-charcoal iron requires a lot more charcoal, too.

T. L. JOSEPH.—A few years ago I gathered operating data from six charcoal furnaces in Michigan, which were making irons running from 1.50 to 2.75 per cent. silicon. The figures that I gave for fuel consumption apply to this range in composition of iron.

Charcoal pig iron is made at lower temperatures than coke pig iron, and therefore less carbon is used. The slag is acid and has a lower free running temperature.

E. J. LOWRY, Chicago, Ill.—The average silicon in 8 or 9 different furnaces in Michigan is about 1.80 or 1.90 per cent. There is not so much difference, although the charcoal furnace does make the grade which has practically no silicon right up to the 2.50 to 2.75 per cent. silicon.

This question of pig iron and the honesty of it revolves practically around the purchaser of castings. The buyer of gray-iron castings buys gray iron principally because he thinks it is cheapest. He wants material that will just stay within a certain definite specification.

You hear that scrap is a detriment to pig iron. I find that as early as 1865 and 1867, in the Lucy and Carrie Furnaces in Pittsburgh, they began using scrap and it has

been a growing practice ever since. I do not believe you can positively trace any defect of pig iron to just the use of scrap; in many cases, especially in the West, I have seen pig iron bettered by its use. No one knows what the effect of scrap is on pig iron, but I do not want its use decreed without a fair chance to learn just what it does and what it really means to the foundryman as well as to the furnace man.

R. MOLDENKE.—We do not know what the effect of charging scrap is. In Germany a commission was appointed to see if its use could not be stopped because they were getting bad castings.

The blast-furnace men say—and rightly, too—“Why don't you foundrymen improve your practice so that you can use this iron?” We would like to save the gray-iron foundry industry by finding some means, not too hard on the blast furnace, of judging, when we get a carload of pig iron, whether we are going to keep it for our best work or only use it for poorer castings.

R. JOB, Montreal, Que.—To maintain uniformity and excellence in any line of manufacture or production it is increasingly necessary carefully to determine the principles on which that particular production depends. The cast-iron foundry is no exception to this rule, and as Dr. Moldenke has pointed out, the ability of the foundry to hold its own and progress depends in no small measure on the care with which these principles are studied.

Basic facts should be determined by systematic cooperative research, just as is being done in other lines of work, and this information should be placed at the disposal of the industry at large. Such information, if given in a clear and practical form, easily understood by the average foundryman, would prove a wonderful help in maintaining and extending the use of cast iron.

We have in mind an instance in which a set of locomotive piston rings was completely worn out after service of 75 miles. Two and three-quarters per cent. of silicon was present and more than 0.75 per cent. of phosphorus, and the grain was coarse and open. Travesties of this kind are unnecessary and would be far less likely to occur if the basic properties necessary for such castings were better understood by the general foundryman. The measurement of spring and of permanent set is easily made, and methods have been developed for the production of iron of resilience and toughness necessary for the service, but the real reasons are not generally understood and are not likely to be for a long time, unless by the concerted cooperative effort of the industry as a whole, as bespoken by Dr. Moldenke.

T. L. JOSEPH.—Charcoal weighs about 20 lb. per bu. and charcoal furnaces operate on from 90 to 100 bu. charcoal per ton of pig iron.

J. S. FERGUSON, Columbus, O.—I operated a charcoal furnace for over 3 years on charcoal made from hard wood, destructive distillation in the retort. Our practice was about 95 bu. and we produced Scotch iron and car-wheel iron, and the weight was from 20 to 21 lb. dry charcoal, about 1900 to 1950 lb. per ton of iron. That was over a period of more than 3 years.

W. B. COLEMAN, Philadelphia, Pa.—I believe a moderate amount of scrap in the blast furnace would not have any very bad effect, just the same as in the cupola. Every iron foundry making cupola castings uses a certain amount of scrap, maybe 40 per cent., maybe 50 per cent. The blast-furnace man may put more scrap into the furnace to save money. Naturally, when melting iron, he does not require as much coke as he would when using ore.

Most of my work has been on the open hearth. I never experienced much trouble on off-irons in connection with open-hearth furnace work, and I have made armor plate and shell steel. The greatest difficulty seems to be in the foundry. You may

have in the open-hearth furnace an iron that is somewhat oxidized and which through the refining operations may be corrected somewhat, but I have yet to learn how to correct an oxidized iron, or whatever you may want to call it, in the cupola. It seems to go right through the cupola and its results go right through the castings.

A company had an oil-barrel type furnace, making very small castings, covers for locks. They were running about 50 per cent. pig iron and of course had a tremendous volume of very light, spindly sticks, so that when they put their pig iron into the furnace they filled it up to the roof with these light gates.

It is very easy to have an oxidizing flame in an oil burner, so they simply drove the flame into the mass; naturally oxidation would take place, and when they poured their castings they were absolutely white on fracturing the iron. If they took the pig iron and used no scrap at all, their castings were gray and under the microscope would show graphite flakes. The silicon was possibly around 2.50 per cent., and the silicon of the oxidized iron about 0.20 per cent. lower. That is not much. A laboratory may vary that much on routine work. The manganese showed a loss of between 0.07 and 0.08 per cent., the percentage content being about 0.50. I think that was the most glaring example I have ever seen of one range going clear over into the other range with very little difference in composition.

In answer to Dr. Moldenke's statement, I would point out that most of the foundries do not have any money to spend, and if they took a lot of odd pieces, put them into a crucible furnace, melted them down and poured them, they would be in a receivership in about a month or so. It is very crude practice to take a chilled pig iron, made at different plants, with different thicknesses of chill and with different water and temperature conditions and try to predict the properties on remelting in cupola.

In order to arrive at some conclusions, I went about 1 in. back from the chill on several pigs, and found I could take a brand of pig iron in the different plants at the same time and they would be somewhat similar, even in that chilled form. However, I did notice the two extremes: iron showing considerable ferrite in the chilled form, and iron in the chilled form showing the phosphor carbide eutectic. You could identify the two extremes and follow them, and after seeing the irons go through the cupola, the results were what one would expect on examination of the pig. That is, the iron would show the absence of ferrite in the pig even though it was chilled, but did not show any ferrite after it went through the cupola. It might be possible that founders could test for the presence of ferrite, in a very crude way, by collecting a button out of a pig and looking at the end of it. I have been doing it.

R. MOLDENKE.—Relative to the method I suggested for testing pig iron: It was not intended for the foundryman to do that but for those who have laboratories to which you could send the chips and have them analyzed. The cost of making the test ought to be no more than the cost of making the analysis.

There is no way that I know of in the foundry of deoxidizing iron when it is once oxidized, simply because the temperatures at which you work in the foundry are not high enough. When you melt steel, you deal with much higher temperatures than you do when you melt iron. The chances of the steel foundrymen getting good results, therefore, are much better than those of the iron foundrymen.

The process used in Germany that gives these very high-strength irons, is this: They have a little air furnace with a shaft attached to it, in which you charge. They have an oil flame—a bad thing, by the way—which blows through the charge and then through the recuperator on top. The air is blown through the recuperator first, giving it 700° F. more than the blast for cupola. The consequence is that they have reached 3000° F. in molten cast iron.

Now as to the question of oil burner: You blow the oil into the whole charge. Oil is a carbohydrate, it is carbon and a lot of hydrogen. You have considerable water, vapor, which, when it touches the iron, oxidizes it. Of course you have oxidized iron.

There is a big difference in actual practice in the use of charcoal iron. I was once called in by a typewriter company on the problem of their carriage frames. They make them of cast iron. I tried everything to meet the conditions to which the machines were subjected. Finally I tried charcoal iron.

The test on the carriage was to put in a bar of iron about 18 in. long and hang a 5-lb. weight on it. If it passed the test of 5 lb. on 18 in. without breaking the casting, they put it through. They wanted something better than that, and I finally succeeded with charcoal iron, in hanging 20 lb. on the bar without breaking the casting. With charcoal iron, of greater purity probably, or with less oxidation, more practical results were obtained. That is why charcoal iron is used today, with the addition of 10 or 15 per cent. to the mixture.

R. H. SWEETSER, Columbus, O.—Dr. Moldenke, when you speak of 65 per cent. scrap, it is 65 per cent. of what? For instance, we will use 4500 lb. of ore mixture, scrap and ore, to make a ton of pig iron. Now, when you speak of 10 per cent. scrap as being allowable, do you mean 10 per cent. of the 4500 lb. of ore mixture or do you mean 10 per cent. of the 2240 lb. of pig iron?

R. MOLDENKE.—I mean 65 per cent. of the actual iron in that ore.

D. J. DEMOREST, Columbus, O.—We have talked about quantity of scrap and not about quality. Twenty-five years ago, when they put scrap in the blast furnace, they did not put in nickel and chrome. You cannot get scrap that does not contain nickel and chrome. Of course, the addition of nickel to cast iron reinforces the effect of silicon, makes the iron graphitize more readily and freeze more slowly and gives a stronger matrix, whereas, the chromium that may be in the scrap does just the opposite. It tends to make it white and less malleable, and I would be inclined to suspect that many of the troubles of the foundryman which result from the use of scrap are not so much because of the quantity of scrap as because of the nature of the scrap.

E. P. ROSS, Riddlesburg, Pa. (written discussion).—Dr. Moldenke's statement, "The quality of pig iron for foundry purposes has deteriorated from the time of cold-blast charcoal pig iron to our present-day foundry iron," is undoubtedly true.

The history of the manufacture of pig iron in the United States, during the past 40 years, has developed one important tendency, namely, to endeavor to satisfy the buyer not only as to the question of quality but as to price as well.

Along about 1880, each pig of iron was struck with a hammer and the sound indicated whether it was No. 1, 2 or 3, etc. A few years later a man who was supposed to know good iron when he saw it took his stand near a pile of freshly broken iron, and the laborers who removed the iron passed near the "grader," who looked at the fracture and told them whether each particular pig belonged to pile No. 1, 2, 3, etc. The foundrymen were able to make good castings when using iron graded in this way; the furnaces were making pig iron largely from virgin ore. Some of the blast-furnace men would scout the idea of using steel or iron scrap in the furnace, claiming it would "burn up." Cast iron scrap was used to some extent and "runner" scrap was put back in the furnace. Probably no one thought this kind of scrap would injure the product. In the early 90's chemical analysis began slowly to determine the grading of pig iron, and at the present time chemical analysis is much the sole factor.

Owing to the absence of absolutely uniform conditions as to the character and quality of raw materials, as well as to operating conditions, the quality of the pig iron produced by a blast furnace is constantly changing, even from hour to hour. Where

foundry pig iron was molded largely in sand molds and allowed to cool slowly, the fracture would open up, depending on temperature and to a large extent on the sulfur content. The fracture method of grading pig iron enabled the "grader" to separate the close-grained iron. One objection to this method would seem to be that good hot iron, high in silicon, would not always give a good fracture, and therefore, it was put in a grade with iron high in sulfur and possibly low in silicon. For this reason among others the method of grading pig iron by means of chemical analysis seems to give the foundryman a better control over his mixture, and enables him to produce a casting to conform to certain specifications.

The pig-casting machine introduced another factor that has more or less of an influence on the quality of the pig iron. In a way it improved the quality, due to the use of the pig-casting ladle that takes the place of a mixer, thus making a cast more uniform in quality. The pigs being cast in cast-iron molds are free from sand and dirt, provided an effort is made to separate the "dross."

The use of cast-iron molds and the sudden quenching of the molten iron destroys the fracture that was used in grading by sight, and in a way may hide the true quality of the product. Apparently it is not known just what influence the practice of suddenly and violently quenching the molten iron (as is the practice followed when using the pig-casting machine) has on the foundry castings or practice. Some foundrymen think that this resultant chill and brittleness given to the pig is carried through the foundry and is shown up in the castings. This may or may not be true.

The pig-casting ladle has introduced another practice into the manufacturing of pig iron that may have had a bad influence on the quality of the product. It is the practice followed by some furnace men of filling the ladle with any kind of miscellaneous scrap that cannot be put through the furnace. The molten iron melts this scrap without any purification taking place, other than to dilute any bad elements that may be in the scrap. So far as we know, this practice is not regular but may be used to clean up unwieldy scrap that has accumulated.

The use of miscellaneous iron and steel scrap, as a blast-furnace raw material, came into general use early in the World War. Shell turnings seem to have been among the first scrap used to a large extent. As the practice of using scrap spread among furnace men the quantity was increased until a large percentage was used, in some instances up to 50 per cent. or more of the ore burden, for short periods of time. The demand for this scrap material increased so rapidly that the material began to become scarce and the price to advance quite rapidly. At first, one could buy this scrap for from \$5 to \$10 per ton, the price finally went to \$20 to \$25 per ton, and no questions were asked as to what elements the scrap contained. The trade names were borings and turnings, machine shop turnings, shovel turnings, etc. The physical qualities of the scrap controlled the amount used; if the material was "bushy" very little could be used, if the scrap could be put in the furnace without clogging the top, more could be used. The larger the percentage used the cheaper the product. The price finally got so high, and the foundrymen began to find so much fault with the pig iron, that the practice of using scrap was more or less discontinued by some producers.

Foundrymen are beginning to realize that chemical analysis, so far as silicon, sulfur, phosphorus and manganese are concerned, does not tell the full story. They have discovered that two metals, with almost identical analyses, often produce widely different results. The presence or absence of certain other chemical constituents, which are now not regularly considered, may have a large influence on the castings. These elements could be introduced by the use of miscellaneous scrap in the blast-furnace burden. There may be a purely physical condition resulting perhaps from various causes.

Some of the causes that may produce these varying results may be classified as follows:

1. Quality of material.
2. Furnace conditions.
 - a. The use of too little coke per ton of iron.
 - b. Over-blowing the furnace with a more or less oxidized product.
 - c. Improper burdening, etc.
3. Harsh treatment of the molten metal by subjecting it to sudden chill in quenching or a combination of the above.

As a general proposition, we think the quality of pig iron has deteriorated and we agree with Dr. Moldenke, as to the need for "research in foundry pig iron."

The furnace problem after all is, as we have intimated, to satisfy the trade. Iron of the best analysis and best quality has no value unless it is marketed at a fair price, yielding a reasonable profit, and in order to accomplish this, we must please the customer. The producer must give his customer iron of the required analysis, but in addition to this the iron must be physically and otherwise of a character and quality to make good castings. The pigs must be clean and free from slag, or anything that looks like slag. They must be of a size convenient to handle and must also be of a design suitable for easy melting.

As Dr. Moldenke says: "The foundrymen should be willing to pay for well made pig iron."

W. G. REICHERT, Elizabethport, N. J. (written discussion).—The paper presented by Dr. Moldenke brings forward the need for research and also some very pertinent questions. It is generally felt that in the last few years there have been great strides for improvement in foundry conditions and also for quality iron. More attention is now being paid by the small concerns, as well as the larger concerns, to the examination of raw material and it is absolutely essential that this be done if the foundry is to live.

Oxidation plays an important part in the iron foundry. All oxidized iron produced in the foundry cannot be laid to the pig iron, but we know that some can, and, if possible, we believe this should be reduced to a minimum. We fully realize that the blast-furnace operators are in a highly competitive business, especially at the present time with the high cost of labor and the comparatively low cost of foreign iron. The question also arises—how can an oxidized pig iron be detected in a quantitative way? This has been one of the problems of the foundrymen. It has been my experience that oxidation cannot be wholly rectified in cupola operations. Statements have been made that oxidation may be detected by a lowering of the total carbon content. It may be lowered under standard conditions of operation while dealing with one iron, but it would not hold true with different irons. A position indicative of oxidation in cupola melting practice may be detected by an excessive drop in the silicon content from that calculated to the actual amount. In the majority of cases the foundryman knows how to correct oxidation in his cupola but he does not know how to overcome oxidation occurring in the pig iron.

The condition also arises in the foundry that with similar analysis and melting conditions the resulting physical properties of the iron are quite different when different grades of pig iron are used. This is especially noticeable in the machining department. Similar analysis and melting conditions will result in the precipitation in one case of graphite, phos eutectic, pearlite and ferrite, and in another case, graphite, phos eutectic and pearlite with no trace of ferrite. Is this condition due to the high scrap used in some blast furnaces, or is it due to the difference in physical structure of sand-cast and quickly chilled iron?

At the Detroit convention of the American Foundrymen's Association a paper was presented by Dr. Piwowarski¹ which points to some of the possibilities of cast iron. It must be conceded that the proper superheating of cast iron is a great stride forward and it is my opinion that we can look forward to even greater possibilities in the future.

We are gradually awakening to the need for research in the foundry and the possibilities in this industry, as greater technical control and research are carried on.

E. TOUCEDA, Albany, N. Y. (written discussion).—The claim is made that the pig iron of today does not square in quality with that made years ago when stack output was not so high. No data have been furnished in support of this claim, it apparently being based for the most part upon the hearsay evidence of founders who have attributed their casting difficulties to some mysterious agency in the pig iron. No data are furnished regarding the extent to which the condition prevails, nor is it indicated in what way castings are not as good as in the old days. The writer was intimately connected with the general foundry industry in those good old days and all one has to do to convince himself that trouble was plentiful is to read the foundry literature published during that period.

Do the tensile properties of castings of the same composition average less now than formerly, and is there at the present time a greater loss from shrink and cracks than occurred then? Or, putting it in another way, do general foundry losses average higher today than 30 years ago? The writer does not think so.

In the paper it is quite positively inferred that pig iron is higher in oxygen and dissolved iron oxide than formerly obtained, an opinion also that is unaccompanied by any data whatsoever. Experiments carried out by the Bureau of Standards show an oxygen content in both coke and charcoal pig iron ranging from 0.01 to 0.06 per cent., the average being about 0.02 per cent., irrespective of type. We must conclude, therefore, that if charcoal pig is superior to coke it is not by virtue of a lower oxygen content.

Quoting in part from the text: "... one very prominent foundryman who took the entire output of two southern cold-blast charcoal furnaces as long as they ran, no matter what the analyses were. By admixing a small percentage of this very high-class pig iron, which did not in the least affect the analyses of the rolls he made, he obtained for his product a much longer wear in service than others obtained who did not so improve their mixtures." In the writer's opinion this statement appears speculative, extravagant and unreasonable. No comparative data are furnished regarding rate of wear, nor any assurance that the rolls were being used under exactly parallel conditions. On the other hand, how exceedingly potent must this charcoal iron have been, when as stated, a small amount added to the mixture could have neutralized the supposed evil effects of the balance of the charge. Certainly, research in this case should have yielded quick returns, for some constituent in very substantial quantity must have been present in the charcoal iron when such a small amount as is inferred could have produced such remarkable results.

Let us take up a few facts regarding malleable iron. It is within the writer's recollection that a positive claim was made by those engaged in this industry that coke iron could not successfully be used for making this product, and without exception all of the manufacturers used charcoal iron. At this time the average tensile properties of the product were less than 40,000 lb. per sq. in. and the average elongation less than 3½ per cent. In due course it was discovered not only that coke iron could be used but that the use of charcoal iron imparted no value whatever to the product as far as could be ascertained.

Let us see what is perfectly feasible practice at the present time, in spite of the alleged deterioration in pig-iron quality and without the use of one pound of charcoal

¹E. Piwowarski: Progress in the Production of High Test Cast Iron. *Trans. Amer. Foundrymen's Assn.* (1926) 34, 914.

iron, on a mixture consisting of 45 per cent. of pig and 55 per cent. of sprue and scrap. Averaging the results of 50 consecutive heats from one particular plant we find the following: Ultimate strength, 58,203 lb. per sq. in.; yield point, 38,085 lb. per sq. in.; elongation in 2 in., 26.88 per cent. I have this concern's assurance not only that it has had no complaints on quality of pig iron, but that it has also been a matter of indifference to them as to what standard brands they use, provided the composition fall within their needed requirements. I would also add that the results quoted would about hold true at this plant if consecutive heats for the last two years were taken.

The members of the organization of which the writer is consulting engineer, with perhaps one or two exceptions, feel satisfied that when they have trouble with their castings it is not the fault of the pig iron. At one time, and it may be true today, one member had a contrary viewpoint due to his thought that a certain brand of pig iron was accountable for an increase in defective castings at his plant. During a discussion of this matter it developed that owing to a much higher freight rate another member who liked the iron actually was paying a premium for it.

It is not to be inferred, that it is the thought of the writer that occasional shipments of poor pig iron which contain dirt and slag are not at times received by the manufacturer, or that occasional shipments are not rejected due to failure to meet composition requirements. He is inclined to believe that Dr. Moldenke has entirely overlooked several vital matters. The founder today is confronted with problems with which the old-timer, lacking experience and facilities, could not have contended with. Today, inspection is extremely rigid, tolerances are in thousandths, and there is little if any margin of profit. That the founder has a great deal of trouble and anxiety under these conditions is hardly a matter of wonder. The foundry superintendent has a heavy load to bear and when things go wrong any excuse or alibi for failure to get results is welcomed, and particularly is this true when the alibi selected is one difficult to refute.

Dr. Moldenke's claim that there has been constant retrogression in the quality of pig iron for foundry purposes, which he has attributed to increase in tonnage, appears strange in view of the following facts, which will be considered with brevity:

In the days when he believes the best pig iron was made, very scant attention was paid to such refinements and details as are in vogue today. Less attention was given to the physical characteristics of the coke; grading and composition-control of the ores had hardly begun; the sizing of the limestone was regarded as of little importance, and furnace tops had not been improved. On the other hand, during the past 15 years there has been a general betterment in furnace design and accessories. But far and beyond all this is the increase in metallurgical knowledge of the present-day furnace superintendent over his old-time predecessor. Can it be possible that these many improvements have gone for naught but tonnage? There might indeed be some reason for Dr. Moldenke's complaint if the larger tonnages were being made under old-time furnace conditions, but tonnage has increased only hand in hand with the improvements that have been outlined, facts that Dr. Moldenke has ignored.

That furnacemen still have their troubles is certain; that occasional poor quality pig is made is likewise true, but that there has been retrogression in the making of pig iron is not believed by the writer.

R. MOLDENKE (author's reply to E. Touceda).—Mr. Touceda is correct in his criticism in that no figures have been given to support the contention that the pig irons of today are not as good as they were formerly, and that a steady retrogression in quality from cold-blast charcoal irons to present-day, high-tonnage coke irons has been manifest. If it were possible to give actual figures, I would not have been urging for 25 years that ways be found to test pig irons for their quality, in addition to their composition. By quality, of course, is meant the physical properties of the cast iron made

under standardized foundry conditions. That tonnage is at the basis of the contention is evident by the abandonment of cold-blast in favor of present-day very hot-blast operation, even on the part of the remaining charcoal furnaces. That there is a quality retrogression from cold-blast charcoal to hot-blast coke iron, well known to everyone conversant with the iron industry, is something I will not waste time and space on in discussion. Mr. Touceda unconsciously admits this when he mentions the very fine malleable cast iron obtained "in spite of the alleged deterioration in pig iron and *without the use of one pound of charcoal iron*" (the italics are mine). I may add that I got even better results than the test bar quoted, over 62,000 lb. per sq. in., and also without charcoal iron; but these fine strengths and elongations are the result of heavy steel additions to cut the total carbon, a practice not known in early malleable days, and which I was first to develop in my plant production work, as long ago as 1895. Furthermore, any reference to malleable pig irons might as well be omitted from this discussion, as these irons are known to be very carefully made, with high coke burdens and an accuracy and care in furnace operation which makes them very desirable even for gray iron production. They are sold for more money than the foundry irons and prove exactly what I have contended right along—that furnaces can make better pig irons than they do today, provided the foundrymen will pay for it.

The imputation of Mr. Touceda that the claim of poorer irons today is based for the most part on hearsay evidence of founders does not accord with actual facts. When the foundryman finds that his losses are gradually increasing, and he has investigated his sand, coke, cores, melting practice and gating of molds, and in desperation he finally changes his pig irons and promptly overcomes his difficulties, you cannot talk of hearsay evidence to him. And of late years this situation has been experienced by not only the smaller foundrymen but by the very large consumers of pig iron. I have found this situation in many foundries when called in to diagnose and correct their difficulties. Unquestionably, the use of high scrap percentages in the furnace burden has accentuated the situation, though judging from what foundry chemists have to say on the subject, some iron districts are freer from this trouble than others.

Referring to the specific instance of the foundryman who took the total output of two southern cold-blast charcoal furnaces for small admixture to his heats for rolls—the foundryman happened to be the late Mr. Seaman of Pittsburgh, Pa., and his works are among the premier roll foundries of America—the performances of these rolls, as also the regular run of custom rolls, are the subject of very careful records on the part of the steel mills to which they went. I often saw these records and discussed them with Mr. Seaman, and well remember his regret when the furnaces in question could no longer get their charcoal supplies and had to close down.

A foundry chemist who goes simply by the analysis he desires, or one who adds to this analysis merely the results of a tensile test and not tests on the resilience of the material produced, has still much to learn on the question of quality in cast iron. With the shrinkage problems introduced by "life-less" molten iron made of oxidized pig irons—or whatever may be the cause of inability to hold liquid metal long enough to feed properly through ordinary gates—more knowledge of the physics of cast iron is required of the foundry chemist, and he and the foundry superintendent and foreman find it a task today to get reasonably sound castings. In overcoming difficulties poorly made pig irons have been found to play a considerable part, and hence the desire to learn more about this end of the situation.

I have freely stated that there has been a great improvement in the uniformity of pig irons as to composition and general furnace practice—doubtless due to better design, handling greater tonnages and a better scientific grounding on the part of the furnace managers—but I still claim that the quality has suffered, mostly through the fact that costs are too high when compared with sales prices, and to muddle through many furnaces have had to use scrap in formerly unheard of proportions. The best

proof—though not in the figures asked for by Mr. Touceda—was given in a recent editorial in *Iron Age*. In commenting on the discussion of this subject before the American Foundrymen's Association, in which the editor in circulating among the furnace-men present learned that though fighting for high quality in their pig irons on the floor of the convention, admission was generally made in the corridors that if the foundrymen wanted better pig irons, they could get quality if they would pay more for it.

If this discussion will hasten the day when pig irons are bought under specification for quality in addition to composition, and the furnacemen will thus command a premium for quality iron, it will have been well worth while.

Carbon Characteristics of Copper-bearing Pig Iron*

BY W. B. COLEMAN,† PHILADELPHIA, PA.

(New York Meeting, February, 1927)

CONSIDERABLE discussion on the effect of iron and steel scrap in blast-furnace burdens was presented in the January, 1927, issue of MINING AND METALLURGY. Therein the question is asked as to what causes the difference in physical properties of pig irons where the chemical analyses are practically the same. This point no doubt has been noticed by a number of furnace operators, but several years ago very little thought was given to it. The foundries purchased pig iron, had it analyzed and found it to be within their specification.

The castings made from this iron at times would be satisfactory and at other times hardness and shrinkage would occur. Since the iron was chemically correct the foundries generally were at a loss as to what to do. They changed their method of melting in the cupola and in some cases this corrected the trouble. In some instances the foundry would continue eliminating a brand of iron until the castings produced were satisfactory. Through this process of elimination they would finally arrive at the conclusion that a certain brand of iron was causing the trouble. During the course of this elimination minor changes might have been made in the melting and in the end satisfactory castings were secured without discovering any definite reason for the trouble. Frequently the foundries blamed the manufacturer of the pig iron although the seat of the trouble was in their own method of melting. I have in mind one foundry in particular that was using three brands of pig iron for producing their castings and their losses were rather high. Three entirely new brands of iron were substituted in the charge and still the losses continued. Even after using six brands of pig iron the management felt that the pig iron was the cause.

The difficulties encountered in the melting of iron in the cupola, over the past several years, reflect in a great many cases on the operation of the cupola. At different times certain brands of iron were unsuitable for producing a certain class of castings, but at the same time one could see no reason why the iron should be considered abnormal.

Irons from furnaces where production and low cost are the main factors that are being used in foundries produce in most cases harder cast-

*Second paper presented at first session of Round Table on Carbon in Pig Iron, February, 1927; for general discussion at second session see p. 482.

† W. B. Coleman & Co., consulting metallurgists.

ings and greater internal shrinkage than irons of the same analysis where the production is not the chief factor.

On a number of occasions during my work as consulting metallurgist, castings have shown internal shrinkage and castings made from the same pattern in the same shop at another time have been perfectly solid.

STRUCTURE AND CHEMICAL ANALYSIS OF CASTINGS

Figs. 1 and 2 are photomicrographs of two castings that show an entirely different structure. The chemical analyses of these particular

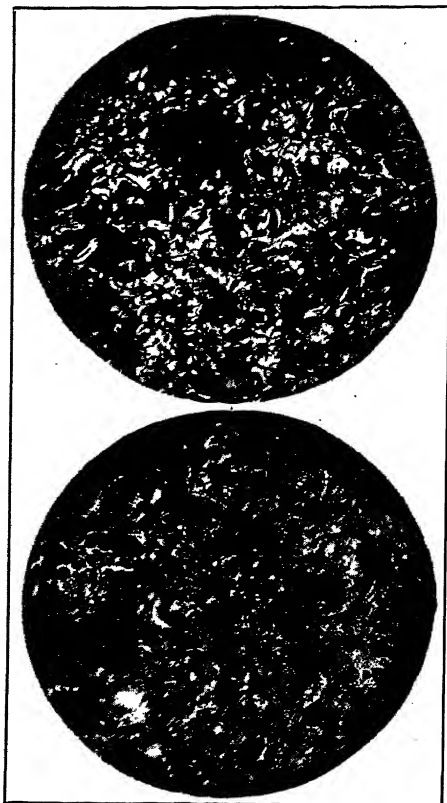


FIG. 1.—STRUCTURE OF A CASTING. ETCHANT, 2 PER CENT. NITRIC ACID. $\times 100$.
FIG. 2.—STRUCTURE OF A CASTING. ETCHANT, 2 PER CENT. NITRIC ACID. $\times 100$

castings, as far as the total carbon, manganese, phosphorus, sulfur and silicon are concerned, are practically the same. On examination it will be observed that one structure has a considerable amount of ferrite pres-

ent and the other structure is entirely pearlitic. The casting that had the ferrite present did not show any internal shrinkage and machined very readily.

For production work in machine shops an iron that has ferrite present will drill or machine faster than one that is entirely pearlitic. Rimbach,¹ says that the "structure of soft easily worked castings must be ferritic." It, therefore, seems desirable for certain classes of foundry work, that is, foundries producing small castings that require a certain amount of machinability, if a ferritic structure can be produced, as time and cost will be lowered in the machine shop.

Many foundrymen in the Philadelphia district felt that pig irons that were copper-bearing produced no end of trouble in the foundry. There is very little information, if any at all, in the literature describing what effect copper might have in pig iron. On many occasions it was found upon analysis that castings considered 100 per cent. perfect as far as machinability and internal shrinkage were concerned, contained copper. These castings were probably made at least several years previous and the structure observed was generally of a ferritic nature. (These statements are confined to small castings running in sections $\frac{1}{8}$ to possibly 1 in. in thickness and not the general run of machinery castings.)

A few experiments were conducted by adding copper in various amounts up to 1 per cent. and the irons were poured into test bars, as well as certain castings. On examining the original iron, which was practically copper-free, very little difference, if any, could be noticed as to hardness, structure and the machinist's opinion on machining castings of one copper content from the other.

TESTS ON COPPER-BEARING IRONS

The Robeson Iron Co., of Robeson, Pa., are producers of a copper-bearing iron for foundry work. In order to determine what the structure might be of the iron as it came from the blast furnace, this concern cooperated in making a large number of tests over a run on foundry iron. Casts were selected at intervals and on each cast arbitration bars were made according to the method set up by the American Foundrymen's Association and, in addition to these test bars, several small castings were made to show the internal shrinkage. Photomicrographs, taken at a point half radius, were made from the test bars.

Tests were taken on a number of casts in order to determine just what the structure of the iron might be, as well as what variances might be noted in the structure from time to time. Fig. 3 shows a representative condition as to analysis and photomicrographs taken on a series of tests while the furnace was producing foundry iron. The majority of the photo-

¹ Rimbach: *Blast Furnace & Steel Plant*. (Dec., 1926.) (Translated from German by Wagner.)

graphs show that there are two kinds of graphite flakes. There are a number of very small graphite flakes scattered through the mass and evidently these small flakes were precipitated after the large ones. It is also noticeable that wherever these very fine graphite flakes occur they are surrounded by ferrite. Ferrite is present in some of the specimens tested in very large amounts. In fact, in several of the specimens ferrite seems to constitute the major part of the surface photographed.



FIG. 3.—TRANSVERSE BAR SECTION. ETCHANT, 2 PER CENT. NITRIC ACID. $\times 100$.

Cast: No. 11,708

Type: Foundry Iron)

Analysis:

	PER CENT.
Combined carbon (color).....	0.19
Combined carbon (combustion).....	0.33
Graphitic carbon.....	3.33
Total carbon.....	3.66
Manganese.....	0.67
Phosphorus.....	0.69
Sulfur.....	0.039
Silicon.....	2.74
Copper.....	0.246
Brinell Hardness:	
Test bar.....	131
Physical Test on Transverse Bar:	
Breaking strength, lb.....	3100
Deflection, in.....	0.14

It was generally observed on the castings made on these different foundry casts that internal shrinkage was minimized, especially where heavy and light sections joined together. The Brinell hardness reading on all of the tests indicated softness as far as this test might indicate. The character of the fracture of the transverse bars showed a mixed coarse crystalline and fine crystalline structure. This variable structure in the foundry irons is due to the precipitation of the finely divided graphite flakes in spots.

TABLE 1.—*Blast-furnace Analyses of Casts and Slags*

Date	Cast							Slag				
	Laboratory No.	Cast No.	Tons	Silicon, Per Cent.	Sulfur, Per Cent.	Phosphorus, Per Cent.	Manganese, Per Cent.	C. C., Per Cent.	G. C., Per Cent.	SiO ₂	Al ₂ O ₃	S
April 8, 1926.....	11,933	480	41.2	3.00	0.027	0.59	0.74	0.27	3.55	34.80	12.00	2.15
April 10, 1926.....	11,944	489	48.2	2.96	0.026	0.70	0.72	0.22	3.57	34.40	12.10	1.90
April 11, 1926.....	11,951	494	41.9	2.32	0.031	0.77	0.70	0.41	3.46	33.60	11.00	2.10
April 12, 1926.....	11,957	499	48.9	3.00	0.030	0.67	0.61	0.36	3.35	33.00	11.90	2.05
April 13, 1926.....	11,977	514	38.8	2.68	0.031	0.70	0.57	0.35	3.50	35.20	11.80	2.05
April 15, 1926.....	11,983	519	47.5	2.73	0.037	0.70	0.70	0.35	3.50	34.00	12.00	2.25
April 16, 1926.....	11,989	524	41.4	2.45	0.032	0.70	0.64	0.35	3.50	34.40	12.30	2.20
April 17, 1926.....	11,996	529	48.3	2.20	0.035	0.68	0.72	0.40	3.40	34.00	11.30	2.20
April 18, 1926.....	11,998	529	48.3	2.71	0.039	0.66	0.67	0.40	3.40	33.71	12.00	2.30
April 20, 1926.....	11,708	539	48.2	2.90	0.030	0.69	0.67	0.40	3.40	34.00	12.10	2.30
April 21, 1926.....	11,719	545	46.1	2.90	0.030	0.69	0.67	0.40	3.40	34.00	12.10	2.30
April 22, 1926.....	11,724	548	40.6	2.64	0.030	0.64	0.64	0.40	3.40	34.00	12.00	2.25
April 23, 1926.....	11,731	555	45.9	2.46	0.046	0.66	0.72	0.40	3.40	35.30	11.80	2.37

Date	Burden, * per Charge										Coke					
	Lime-stone	1	2	3	4	5	6	7	8	9	Total Ore Burden	Blast, Cu. Ft. per Min.	Blast, Cu. Ft. per Lb.	Average Blast Temperature, Deg. F.	Hearth Area, Lb. Coke per Sq. Ft. per Min.	Tuyere Area, Cu. Ft. Blast per Sq. In. per Min.
April 8, 1926.....	9,400	3,600	4,400	3,000	350	500	1,000	4,800	540	15,050	25,000	59	1,175	3.53	115	
April 10, 1926.....	9,400	3,600	4,400	3,000	350	500	1,000	5,300	20	15,070	25,000	59	1,170	3.53	115	
April 11, 1926.....	9,400	3,600	4,400	3,000	350	500	1,000	5,300	17	15,067	25,000	59	1,160	3.70	115	
April 12, 1926.....	9,400	3,600	4,400	3,000	350	500	1,200	5,800	15	15,065	25,000	57	1,180	3.64	115	
April 15, 1926.....	9,400	3,615	300	4,400	3,200	350	1,300	5,300	900	15,350	25,000	57	1,190	3.64	115	
April 16, 1926.....	9,400	3,660	300	4,400	3,300	375	1,300	5,300	875	15,475	25,000	58	1,200	3.59	115	
April 17, 1926.....	9,400	3,655	300	4,400	3,300	375	1,300	5,300	875	15,475	25,000	58	1,200	3.59	115	
April 18, 1926.....	9,400	3,655	300	4,400	3,300	375	1,300	5,300	900	15,475	25,000	58	1,220	3.70	115	
April 20, 1926.....	9,400	3,655	300	4,400	3,300	350	1,300	5,300	900	15,450	25,000	61	1,220	3.42	115	
April 21, 1926.....	9,400	3,655	300	4,400	3,300	350	1,300	5,300	900	15,450	25,000	64	1,220	3.25	115	
April 22, 1926.....	9,400	3,655	300	4,400	3,300	350	1,300	5,300	900	15,450	25,000	62	1,200	3.37	115	
April 23, 1926.....	9,400	3,710	300	4,400	3,300	350	1,300	5,300	900	15,450	25,000	59	1,100	3.53	115	

EFFECT OF REMELTING

In order to ascertain just what effect remelting these irons in a cupola might have on the structures, iron was taken from these particular casts

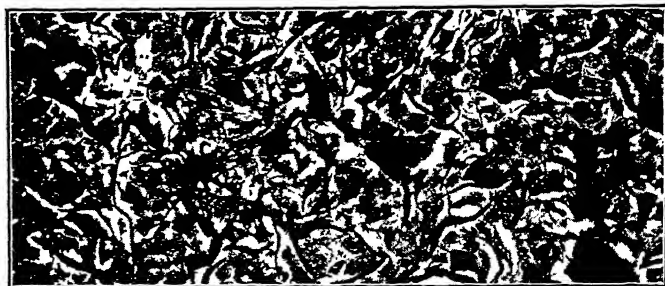


FIG. 4.—TRANSVERSE BAR STRUCTURE. ETCHANT, 2 PER CENT. NITRIC ACID
× 100.

Material: Cast Iron. Photomicrograph and drillings for chemical analysis.
from Transverse Test Bar.

Mixture: Robesonia Pig Iron, 60 per cent.; scrap, 40 per cent.

Type of Castings Produced: Thin hardware parts, such as lock plates, etc.

Analysis:

	PER CENT.
Combined carbon.....	0.52
Graphitic carbon.....	2.96
Manganese.....	0.54
Phosphorus.....	0.551
Sulfur.....	0.077
Silicon.....	2.57
Copper.....	0.10
Brinell Hardness:	137
Physical Test on Transverse Bar:	
Breaking strength, lb.....	2800
Deflection, in.....	0.100

* The materials used in the burden are numbered 1 to 9. The following is a description of the material used: (1) Heating furnace cinder; lumpy cinder, containing about 62 per cent. iron and produced by heating furnaces of American Bridge Co., Pencoyd, Pa. (2) Roasted Cornwall ore; largely self-descriptive. Lumpy portion of Cornwall ore is roasted in old-fashioned Giers kilns and the finer portion sintered in Greenawalt sintering pans. Roasted lump ore contains approximately 45 per cent. iron and 0.50 per cent. sulfur. (3) Cornwall sinter; the finer portion of the Cornwall ore sintered in Greenawalt pans; also contains about 46 per cent. of iron. The sulfur is lower than in the roasted ore, running 0.20 per cent. or under. (4) Lavino slag; slag obtained from a blast furnace running on ferromanganese; contains on an average about 10 per cent. of manganese, and is used to add this element to the charge. (5) Lebanon scale; obtained from the Lebanon Iron Co.; a product of the rolling of wrought iron. Iron, 60 to 65 per cent. (6) Lebanon cinder; granulated, puddled furnace cinder, containing about 55 per cent. of iron. (7) Richard ore; a phosphoric New Jersey magnetite, containing about 58 per cent. of iron and about 0.80 per cent. phosphorus. (8) Phosphate rock; self-descriptive, and is used to bring up the percentage of phosphorus in foundry iron when that element is otherwise deficient. It contains 14 per cent. of phosphorus, the balance being principally lime. (9) Furnace scrap; obtained from the furnace runners, and from droppings of the pig machine, etc.

and remelted in a cupola using 60 per cent. pig iron and 40 per cent. scrap. The scrap used was the shop product and contained a certain amount of the copper-bearing iron that was being used previous to the tests conducted. The castings produced in this particular shop were very thin hardware parts and considerable drilling and tapping were necessary in producing a finished product.

Fig. 4 is one of many tests and shows a representative condition produced in a foundry using this particular grade of pig iron in the cupola. We observed in these photographs that ferrite exists in every case, showing that it is possible to maintain the original structure of the pig iron after it has been remelted in the cupola. These bars were produced under normal conditions, no effort being made to correct any metallurgical problems in their cupola practice, if necessary. The castings produced from these heats were very satisfactory when subjected to machining operations.

BLAST-FURNACE DATA AND OBSERVATIONS²

The following are some of the general lines of the Robesonia furnace: Hearth, 12 ft., 4 in.; bosh, 18 ft.; height, 80 ft.; stock line, 12 ft., 4 in.; volume to stock line, 13,000 cu. ft. Although the Robesonia furnace has been in existence for many years, a number of changes have been made to bring the furnace up to the present modern trend of blast-furnace operation.

The average yield of pig iron is 49 per cent. of the total ore charge. Sufficient flux is used in the operation of the furnace to produce properly smelted iron and 1700 lb. of slag is produced per ton of pig iron.

During the run on foundry iron described in this paper, the blast pressure went up to as high as 20 lb. several times. Slight slips would then occur and at this point an examination of the iron would not reveal any decided change in structure.

In Table 1 the blast-furnace analyses of the casts are shown, as well as the slag analyses. The analyses accompanying Figs. 3 and 4 were made on the transverse test bar that was being tested and explain slight variations arising on comparison with the blast-furnace analyses. The table shows the make-up of the burden, together with the temperatures and volume of blast delivered per minute. The cubic feet of blast per minute are taken from the piston displacement, and since engines of very old make are used at the furnace, it is possible that the figures may not be very accurate.

²Submitted by Mr. Crewe of the Robesonia Iron Co.

ANALYSES AND STRUCTURES OF OLD CASTINGS

Of possible interest to metallurgists in general are the analyses and structures of some iron castings made no doubt from direct metal years ago. Dr. Mercer, of Doylestown, Pa., a man who has one of the largest



FIG. 5.—“JUDGE NOT PLATE,” READING FURNACE PLATE, CAST 1756. ETCHANT, 2 PER CENT. NITRIC ACID. $\times 100$.

FIG. 6.—“JUDGE NOT PLATE,” READING FURNACE PLATE, CAST 1756. ETCHANT, 2 PER CENT. NITRIC ACID. $\times 500$.

collections of antiques and old stoves in the country, very kindly submitted corners broken from two plates that he had in his museum. He felt that it would not only be of interest to him, but also to furnacemen and foundrymen to know just what constituted these plates. The “Judge Not Plate,” made at the Redding furnace and cast in 1756, is

approximately 2 ft. square and about $\frac{5}{8}$ in. thick. The other plate was cast at the Durham furnace about 1790 and is roughly 2 ft. square with a thickness of $\frac{3}{8}$ to $\frac{1}{2}$ in.

It is very interesting to note that the structure of these irons is entirely pearlitic and the graphite flakes are very large, well-formed and evenly

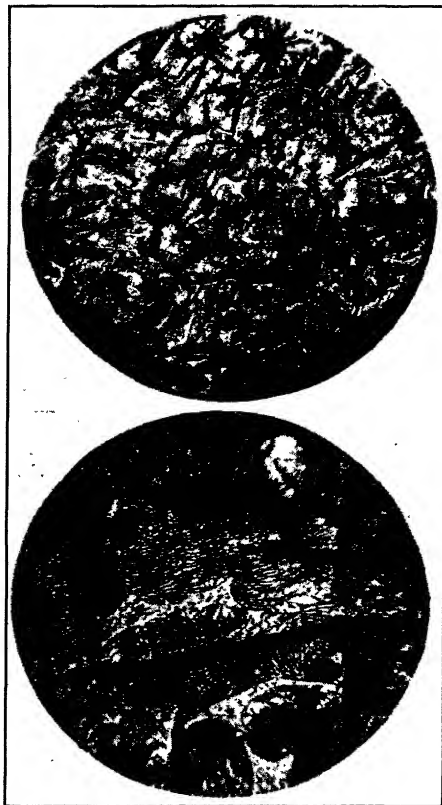


FIG. 7.—DURHAM FURNACE PLATE, CAST 1787–1791. ETCHANT, 2 PER CENT. NITRIC ACID. $\times 100$.

FIG. 8.—DURHAM FURNACE PLATE, CAST 1787–1791. ETCHANT, 2 PER CENT. NITRIC ACID. $\times 500$.

distributed. The general character of the fracture of these castings is gray, very similar to a plate of the same dimensions with a silicon content of 2.50 per cent., if made today.

Corners of these plates could be cut with a knife, so exceedingly soft is the metal. This is remarkable when you observe that the silicon

in both of these plates is well under 1 per cent. Following are the analyses of the two plates (Fig. 5, 6, 7 and 8):

	"JUDGE NOT PLATE" REDDING FURNACE PLATE CAST 1756	DURHAM FURNACE PLATE CAST ABOUT 1790
Combined carbon, per cent.....	0.89	0.66
Graphitic carbon, per cent.....	2.60	2.96
Manganese, per cent.....	0.05	0.24
Phosphorus, per cent.....	0.084	0.46
Sulfur, per cent.....	0.029	0.03
Silicon, per cent.....	0.33	0.70
Copper, per cent.....	0.003	0.001
Nickel, percent.....	Nil	Nil
Chromium, per cent.....	Nil	Nil

SUMMARY

1. Pig irons have been produced of approximately the same chemical analysis, but set forth entirely different physical properties when remelted and poured into castings.

2. A pig iron on being remelted and tending to produce some ferrite in the castings, promotes greater machinability and softness.

3. The additions of less than 1 per cent. metallic copper to molten iron did not seem to alter the physical properties of the castings and tests produced.

4. A blast furnace operating on a burden carrying a copper-bearing ore can produce a grade of pig iron that is highly adapted for the manufacture of small castings where production machine methods are required.

5. Such pig iron shows a low Brinell hardness test, good transverse and deflection tests, and high total carbon.

DISCUSSION

H. M. BOYLSTON, Cleveland, O.—Does the author wish us to infer that it does not make much difference what the structure of the iron is, in its effect on the physical properties, so long as there is some free ferrite?

W. B. COLEMAN.—No, I did not want you to draw that inference because I mentioned that I was dealing with castings of very small size, where machinability was desired. I believe that where you are pushing castings through the machine shops, in production work—where you have free ferrite—yes, but if you are dealing with ferrite iron in the making of large castings, that is another matter. I do not know how to change the pearlitic iron over to the ferrite iron in the cupola.

A. MARKS, Birmingham, Eng.—We have always found that additions of copper were quite useless in foundry work and their influence can be entirely mitigated by the condition of the carbon. The same applies to additions of nickel and chromium. An immense amount of research work has been done in both Great Britain and Germany, and you are no doubt aware of the influence of these elements in connection with foundry work. It has been quite well established that for machine practice one can practically eliminate the addition of these elements.

We have a Cast Iron Research Association which was originally started by the Institute of British Foundrymen, and in it both blast-furnace manufacturers and foundrymen are cooperating to form a joint research association, with Mr. Pierce as the director and Mr. Fletcher as metallurgical consultant, both of them very able men, and today we are making a considerable number of advanced researches.

The big concerns are also carrying on an enormous amount of independent research work and we can get, and have gotten, in Great Britain commercial castings on the order of 22 tons tensile strength, which are of very high impact test and quite independent of the addition of any elements such as nickel or copper. The solubility of copper in the ferrite is so low that its influence is negligible. It begins to separate out before we get to 1 per cent.

The advances which are going to be made by that research association I believe could be well copied in this country. Perhaps Dr. Moldenke was thinking along those lines when he spoke of the need for research, but in addition to that combined research there is an enormous amount of independent research work being done by private foundries. As our association numbered among its members some of the largest foundries and certainly some of the largest diesel engine producers in the world, we were particularly interested in diesel castings and turbine castings. During the war we had to take any pig iron; we not only had to take pig iron, but we had to make all our castings of any scrap, and we could produce in the cupola as good castings from any class of scrap, provided the phosphorus was down, as could be made with the best charcoal pig irons. That was possible by paying attention to the structure of the material, the temperature of the mold at the time of casting, etc.

The average British Admiralty tensile strength in 1914 was 11 tons per sq. in. Today we can produce commercial castings with anything from 20 to 22 tons per sq. in. The only question which arises is: will the machine shop pay for the cost of production? Usually, they will not pay for that cost. For instance, you can give the machine shop wearing surface on an automobile casting, a cylinder which will wear indefinitely, or a piston ring which will wear indefinitely, but the problem is: will it pay for itself? Of course, we tend to move in that direction in Great Britain, toward making an article which will wear forever, but at the same time over there we are confronted with this question of cost.

By using suitable mixtures and a proper mold of definite temperature, we can use any brand of low-phosphorus pig iron and get any strength up to 22 tons to the sq. in. on commercial castings, but there are very few automobile concerns in the world that would pay the increased cost for that cylinder-block.

I think that if you can do anything in the way of combined research, such as we are doing, you will educate the foundry people so that they can use any class of pig iron, and the tendency to pass the responsibility from the foundry to the pig iron manufacturer will very shortly be eliminated. Today, of course, we tend to throw the onus of the trouble in the foundry on to the foundryman, the molder, but even there we are getting this, as metallurgists. I, as a metallurgist, was at the International Exhibition in Great Britain, having been appointed as one of the judges on the molding work, so you see we tend not only to make our metallurgist an engineer but also a molding man, in order that he may appreciate every side of the question.

In that connection, when the Armistice was declared it was decided to get a single blast-furnace burden for special diesel work, purely and simply to save us trouble, to save us from recurring difficulties in our foundries and to make the thing more automatic. We had two blast furnaces running on a low-phosphorus iron with a definite lower limit or definite maximum limit of graphite specified, and from that we made all our diesel castings, such as are used in the big diesel motor lines, and we had no further trouble.

R. MOLDENKE, Watchung, N. J.—You will find it very hard to convince any foundryman here that he wants to specify graphite only for carbon in his pig iron.

I am glad you find that the copper has no effect at all. The belief has always been that copper acts like sulfur, that the copper hardens the iron. One of the eastern Pennsylvania blast-furnace men told me that copper in the iron means a low volumetric sulfur. He said he had so much copper in his iron that where he got 0.02 per cent. in the volumetric analysis, it was 0.08 per cent. in the gravimetric analysis. Perhaps the real reason you blamed the copper was because you thought you had 0.02 when you really had 0.08.

When Dr. Piwowarsky was here recently he presented his theory of these tremendously strong irons, as follows: "The theory of crystallization in general is that there are present in the solutions or the liquids (whatever you have to crystallize) invisible points which start crystallization. Those invisible points, when they begin to come together so they can just be seen, are called nuclei, and when these nuclei are present the crystal starts." Now, the claim of Dr. Piwowarsky has been this: That when you take any iron and melt it, there will remain some graphite undissolved, and the hotter you melt, the less there is. But in ordinary practice there are enough of these little particles of colloidal graphite present that when the metal begins to cool a little, the crystal starts to grow and you get a long crystal of graphite and weak iron. If you can raise the temperature of the melt, everything is wiped out; you have a complete solution without any graphite to start with. Then the metal can cool down to far below what it will crystallize at, and when it does you get a strong piece of metal. The fact is, that the higher you can bring up the superheating point, even if you let it cool afterwards, before you pour it, the stronger will be your material.

A Pig Iron, Low in Total Carbon, is in Demand for Use in Various Industries*

By ENRIQUE TOUCEDA,† ALBANY, N. Y.

(New York Meeting, February, 1927)

THE question as to the proper amount of total carbon that the malleable founder would prefer to have in pig iron for use in the production of air-furnace, white iron castings, must be considered from different angles, that is, whether it is desirable or advantageous from an economic standpoint to use some kind of low-carbon scrap on his mixture, or make use of pig and sprue only. Whereas silicon is the acknowledged scrap carrier in the case of cupola gray iron castings, carbon performs this role for the most part in malleable iron practice, irrespective of the character of furnace used for the production of the white iron castings. In this particular, the advantage lies with the maker of gray iron castings in that the silicon content in pig iron through a regulation of furnace temperature is under very easy control by the blast-furnace manager, whereas he has little or no control over the resultant percentage of carbon, as this is dependent on the quantity of coke in the hearth, and hearth temperature, coupled with the amount of hearth slag and its particular composition.

As perhaps more than 95 per cent. of white iron castings are made in air furnaces the following statements are intended to cover that situation particularly:

In the malleable process the composition of the hard iron castings (that is determined upon as best suited for the work in hand) is the one factor that must control the composition of the mixture to be used; because if the elements are not between certain definite limits individually and collectively the annealed or finished product will be unsuitable, due, perhaps, to lack of sufficient strength or because it is too hard, to machine easily. The annealed product can be divided into two classes, one of which greatly predominates, that is, by far the largest tonnage is used for machine parts where high strength and ductility are essential requisites, while a far smaller amount, light in weight and thin in section, is used for other manufactured products in which castings of any reasonable degree of strength and ductility have proved satisfactory.

* Third paper presented at first session of Round Table on Carbon in Pig Iron, February, 1927; for general discussion at second session see p. 482.

† Chemist and metallurgist.

When comparatively high strength and ductility are essential characteristics, as usually is the case, the white iron casting should not have a carbon content in excess of 2.50 per cent., in order that the carbon in the annealed casting will be sufficiently low, while for higher strengths a total carbon content considerably lower than 2.50 per cent. is more desirable. Assuming that the ratio of pig to remelt (sprue) in the regular mixture is 60:40; that the pig iron will average 3.75 per cent. in total carbon; that the average loss in carbon in air-furnace practice will average 0.55 per cent. actual, and that the percentage of carbon desired in the white iron castings is 2.50, it is manifest that if the conditions cited are fulfilled a white iron will be obtained which will be considerably higher in carbon than this amount. In short, if such pig iron is to be used, and a strong casting is desired, less than 60 per cent. of pig must be incorporated in the mixture and a certain percentage of steel or annealed malleable iron, or both substituted in its place in order that the carbon content of the white iron castings will not exceed 2.50 per cent. Or it might be possible through the use of iron ore, mill scale, etc., to lower the carbon to the desired limit instead of using steel scrap, or again a combination of the two methods can be employed. The main objections however to the use of steel scrap in the mixture lie in the fact that it is very liable to carry an objectionable amount of chromium or other undesirable impurities. To a lesser extent this obtains also in the case of miscellaneous malleable iron scrap.

If the foregoing is true it is at once apparent that even if carbon in pig iron is considered in the light of a scrap carrier, no possible advantage arises from the use of high-carbon pig iron, first because the use of the mixture of steel or malleable scrap introduces a serious uncertainty regarding the objectionable impurities these products may carry, and second because at times there is very little if any difference in cost between scrap and pig iron. Consequently, the only logical conclusion at which one can arrive is that it would be greatly to the advantage of the manufacturer of air-furnace castings that subsequently are to be annealed if he could purchase the bulk of his pig having a carbon content not in excess of 3 per cent., while if it was lower no damage would result as there would be no difficulty in raising the carbon through the use of higher carbon iron. Stated in another way, if a 3 per cent. carbon pig could be purchased to form the base of the mixture, mixture control as far as carbon is concerned would be facilitated to a very large extent, while the use of such iron would serve to eliminate many of the difficulties that exist today, making possible final adjustment through the use of a minimum amount of additions whether the latter are for the purpose of raising or lowering the carbon. It might be pointed out that air-furnace temperatures cannot be so manipulated that the carbon can be oxidized preferentially to the silicon, as is possible in some of the other metallurgical processes.

The writer recognizes the fact that the making of a pig iron running 3 per cent. or lower in carbon regularly cannot be effected in the blast furnace proper, but he does believe it would be possible and practical to rig up to do this through an auxiliary installation in the cast house. Finally, it can be stated that it would prove a great advantage to the malleable iron founder if his adjustment of carbon did not necessitate the use of steel scrap. Also, if proper thought is given the matter it will be realized that in practically all ferrous metallurgical processes a low-carbon pig would be more suitable than one that was high in this element.

DISCUSSION

R. MOLDENKE, Watchung, N. J.—I object seriously to the assertion that we want low total carbon out of the blast furnace. It was previously stated that if the scrap were put into a blast furnace and it filled up with iron, more scrap could be used up in that way. The foundryman wants to use scrap in his mixture his own way. We want a high total carbon pig iron which we can dilute with steel and get low total carbon. I think Professor Touceda is afraid that when he buys steel scrap in the market, he gets a lot of chromium in it. I want high total carbon in pig iron.

R. H. SWEETSER, Columbus, O.—Everyone recognizes that these users want to do their own mixing. We have been surprised at some of the statements that have been made by the steel men, malleable iron men and foundrymen as to the amount of scrap they want to use with the pig iron. Now, do the users of pig iron really want a large amount of carbon and in what form do they want it?

A. MARKS, Birmingham, Eng.—There are two points of view from which to view this question. I was consultant to one of the British malleable pig iron manufacturers in England, and my home overlooks the first malleable iron blast furnace put up in England. For malleable iron practice and for gray iron practice, we run low-carbon irons, but with the difference that the minimum we specify for ordinary gray iron will be on the order of 3.25 per cent. carbon. There are certain objections to adding steel, and adding it in excess, but provided the silicon is high enough, any amount of steel may be added in the cupola.

So in thinking of the production of iron, one has to differentiate between production for the gray iron and the malleable iron, and certainly there is a demand, in Great Britain at least, for special irons for malleable work and also special irons for chilled roll manufacture, although even there we make chilled rolls only from runs of what you would term "cheap" pig irons. We make chilled roll runs in cupola practice as well as open-hearth practice. So you will find that the chief demand for these low carbon irons is in the malleable industry.

J. H. SCHMUCK, Naugatuck, Conn.—Speaking from the standpoint of the producer of malleable castings, we do not feel at the present time that low-carbon pig iron would be particularly desirable. In such a highly competitive industry as that of producing malleable castings material costs are of the greatest importance. We do not believe the blast-furnace men could make pig iron, as Mr. Touceda specifies in this paper, any more cheaply than they are making malleable pig as now produced. As Mr. Touceda says, the pig iron in the air furnace charge has to be diluted with a certain percentage of steel or annealed malleable iron, malleable scrap, so called. However, it is unnecessary to use steel scrap although from reading this paper one would be led to assume that in malleable foundries the use of steel is almost a necessity. A very large tonnage

of malleable is now made from either pig iron and remelt alone or from pig iron, malleable scrap and remelt.

Further, he says that almost all of the scrap used now is liable to be contaminated with chromium or some of the elements used in alloy steels. It is unnecessary to use scrap of that sort. Rail scrap is very satisfactory for the malleable producer, as also is some structural scrap, which does not contain elements that are detrimental to malleable iron. Hence, from the standpoint of the producer, we do not feel anything would be gained by using a low-carbon pig iron.

J. T. MacKENZIE, Birmingham, Ala.—If you could get a 3.50 to 3.60 per cent. carbon in the foundry, which would eliminate the use of steel to some extent, it would relieve the uncertainty as to the amount of carbon picked up by the melt in going through the cupola. If you are forced to use a high-carbon pig iron say, of 4 per cent., and reduce it to 3.50 per cent. with steel, and anything happens to the cupola so that the iron runs cold for a short time, you will find that the cold iron is very low in carbon because the carbon absorption of the steel is to a large extent a function of the temperature. Thus, instead of having a 3.50 per cent. in your mix at the very time you need the fluidity with your cold iron, you lose it by its probably coming out 3.30 per cent. There is a large block of the pig-iron people who would be thoroughly in sympathy with Mr. Touceda's suggestion to produce pig iron of 3.50 per cent. carbon.

J. H. SCHMUCK.—Mr. MacKenzie's remarks hold to a certain extent in the instance of the malleable industry as applied to the producer of the cupola malleable, but would not hold with the producer of air-furnace malleable.

Carbon in Pig Iron*

By RALPH H. SWEETSER,† COLUMBUS, OHIO

(New York Meeting, February, 1927)

CARBON in pig iron is not only essential but, ordinarily, it is the most abundant metalloid present; iron without carbon could not be pig iron. Carbon in pig iron has been accepted, but seldom specified. How it gets into the pig iron as it is being smelted in the blast furnace, and why just about so much of it enters into the pig iron in certain blast furnaces, and more of it, or less of it, enters the pig iron in other blast furnaces, or in the same blast furnace under varying conditions, are problems that have not yet been satisfactorily solved.

During my own practical blast-furnace experience, I collected analyses of pig iron, including the graphitic and combined carbon, and recorded them in reference to the percentage of silicon present, having in mind the generally accepted explanation that "silicon drove out the carbon." To this actual experience, I have added some analyses given to me by J. B. Rogers, who was superintendent of the blast furnaces of the Ashland Iron & Mining Co., Ashland, Ky., and was making "silvery iron" at the time. These analyses are shown in Table 1.

The analyses do not tell the whole story, but they show that with Pocahontas beehive coke, the total carbon was generally high. The data relating to the composition of the slag are lacking but my own customary practice was to have a rather basic slag.

During my experience in making pig iron for the Bessemer and open-hearth departments, there were times when the steel men would complain about some mysterious quality in the pig iron that caused bad steel, in spite of the fact that the percentages of silicon, sulfur, phosphorus and manganese were within the specifications. Many reasons were advanced for this undetermined "poison" in the pig iron, but the favorite one offered by the steel men was that the blast-furnace man had "oxidized" the pig iron at the tuyeres in his eagerness to get big production and low cost.

To these complaints of the users of steel-making pig irons were added those of the gray iron-foundry men and the malleable iron men. These groups are unanimous in their conclusions that the trouble lies with the blast-furnace men; but there is not yet any harmonious conclusion as to the exact cause and remedy.

*Last paper presented at first session of Round Table on Carbon in Pig Iron, February, 1927; for general discussion at second session see p. 482.

† The American Rolling Mill Co.

TABLE 1.—Analyses of Pig Iron Arranged According to Percentage of Silicon

Iron	Silicon, Per Cent.	Sulfur, Per Cent.	Phos- phorus, Per Cent.	Manga- nese, Per Cent.	Graphite Carbon, Per Cent.	Combined Carbon, Per Cent.	Total Carbon, Per Cent.	Total Metals, Per Cent.	Coke
Basic.....	0.26	0.022	0.153	0.80			4.40	5.74	Kanawha and Pocahontas
Low phosphorus.....	0.47	0.039	0.025	0.31	3.55	0.83	4.38	5.22	Pocahontas
Low phosphorus.....	0.78	0.020	0.028	0.40	4.27	0.46	4.73	5.05	Pocahontas
Low phosphorus.....	0.85	0.015	0.028	0.48	4.80	0.55	4.94	6.31	Pocahontas
Low phosphorus.....	1.03	0.010	0.029	0.55	4.30	0.40	4.70	6.32	Pocahontas
Malleable.....	1.04	0.028	0.169	0.69	3.60	0.90	4.50	6.43	Pocahontas
Low phosphorus.....	1.10	0.013	0.163	0.87	3.64	0.88	4.52	6.06	Kanawha and Pocahontas
Malleable.....	1.29	0.015	0.026	0.54	4.32	0.64	4.96	6.83	Pocahontas
Malleable.....	1.33	0.018	0.183	0.66	3.65	0.80	4.45	6.04	Kanawha and Pocahontas
Low phosphorus.....	1.58	0.008	0.028	0.54	4.25	0.35	4.60	6.76	Pocahontas
Low phosphorus.....	1.87	0.012	0.028	0.59	4.44	0.21	4.65	6.95	Pocahontas
Foundry 2x.....	1.70	0.023	0.201	0.176	3.10	0.19	3.29	5.49	Broadtop
Bessemer.....	1.85	0.021	0.058	0.38	3.50	0.75	4.25	6.56	Pocahontas
Low phosphorus.....	1.94	0.014	0.032	0.60	4.20	0.21	4.41	7.00	Pocahontas
Foundry —2x.....	1.98	0.015	0.144	0.173	3.23	1.15	4.38	6.69	Broadtop
Low phosphorus.....	2.00	0.010	0.029	0.60	4.43	0.15	4.58	7.22	Pocahontas
Low phosphorus.....	2.19	0.012	0.032	0.63	4.40	0.15	4.55	7.42	Pocahontas
Low phosphorus.....	2.46	0.011	0.030	0.69	4.32	0.12	4.44	7.64	Pocahontas
Foundry —2 Plain.....	2.49	0.050	0.425	0.115	3.40	0.12	3.52	6.60	Broadtop
Foundry 1 Scotch.....	2.79	0.017	0.559	0.173	3.14	0.74	3.88	7.42	Broadtop
Foundry 1 Scotch.....	3.09	0.008	0.473	0.216	3.83	0.24	4.07	7.92	Broadtop
Foundry 1 Scotch.....	4.01	0.053	0.533	0.227	3.29	0.41	3.70	8.23	Broadtop
Low phosphorus.....	4.69	0.005	0.024	0.70	3.73	0.09	3.82	9.34	Pocahontas
Silvery.....	5.12	0.034	0.534	0.315	3.22	0.14	3.36	9.36	Broadtop
Silvery.....	7.69	0.024	0.544	0.302	2.21	0.50	2.71	11.27	Broadtop
Silvery*.....	5.60	0.031	0.149	1.15	2.42	0.58	3.00	11.27	Unknown
Silvery.....	10.34	Trace	0.54	0.54	1.92	0.07	1.97	13.41	Unknown
Silvery*.....	11.26	0.020	0.84	0.84	1.52	0.03	1.55	14.51	Unknown
Silvery*.....	12.19	Trace	0.48	0.76	1.52	0.06	1.58	15.01	Unknown

* Ashland, Ky.

For the past four years at our annual meeting, I have asked the open-hearth men, "How much carbon do you want in your pig iron, and why?" This question is still unanswered, except for the counter question, "If we say how much carbon we want, can the blast-furnace man control the percentage?"

BLAST-FURNACE RESEARCH

With the aim of finding out, if possible, how to control the carbon that goes into the pig iron and to determine the connection between the analysis of the gas at the tuyeres, the analysis of the slag and pig iron and the temperature of the slag and pig iron, and further to determine if possible that part of the blast furnace where the different metalloids enter the pig iron, The American Rolling Mill Co. made preparations late in 1922 to conduct tests at their Columbus blast furnaces. The greatest difficulty was to obtain pyrometers that would give the temperatures of molten pig iron and slag. Optical pyrometers were not reliable because of the fumes and flames, but were used part of the time, as a check, when the surface of the iron was entirely clear. Finally our research department and Prof. D. J. Demorest, of Ohio State University, devised a thermocouple pyrometer protected by a fused silica tube and an outer tube of carbon.

Continuous observations were made at East furnace, Columbus Works, for the 5 days beginning with the day turn of May 3, 1923. S. P. Kinney of the U. S. Bureau of Mines made the investigation of the gas analysis at the tuyere level and the results have been presented in one or more reports of the Bureau of Mines, and also published in the *Transactions*.¹ No internal temperatures at the tuyere level were observed.

The temperature of the molten iron as it flowed from the furnace was observed every 30 sec. by means of a 60-in. platinum-rhodium thermocouple (0.020 in. dia.) encased in fused silica protection tubes with closed end. The outside protection was a 24 by 2-in. Acheson graphite electrode bored out on a lathe so as to take the silica tubes with the thermocouple inside. The lead wires were connected with a Leeds and Northrup potentiometer indicator.

The pyrometer was firmly placed so that the protected thermocouple was immersed in the molten iron between the skimmer and the dam. Of course there was a lag in the recorded reading on account of the several protections of the thermocouple. This lag was checked with an optical pyrometer which could be used on the clear surface of the iron between the skimmer and the dam. All temperatures are in degrees Fahrenheit.

¹ G. St. J. Perrott and S. P. Kinney: Combustion of Coke in Blast-furnace Hearth. *Trans.* (1923) 69, 543.

The record of Cast No. 2457 at East furnace, at 11 a. m., May 3, 1923, was as follows:

RECORD OF CAST No. 2457, EAST FURNACE, MAY 3, 1923

Time			Iron Sample, No.	Temperature Optical, Deg. F.	Temperature Thermocouple, Deg. F.	Slag Temperature, Deg. F.
Hr.	Min.	Sec.				
10	47	50	1	2418 2450 2470 2500 2506	1986 2061 2370 2483 2514	Slag starts
	48	20				
		50				
	49	20				
		50				
	50	20	2	2512	2538	
		50				
	51	20	3	2512	2538	
		50				
	52	20		2530	2544	
		50				
	53	20	4	2578 2578 2578 2578 2572	2556 2562 2562 2569 2575	Slag reaches tube 418 808 1207 1526
		50				
	54	20				
		50				
	55	20				
		50	5	2578 2566 2572 2572 2566	2575 2575 2575 2575 2550	1816 2043 2212 2313 2370
	56	20				
		50				
	57	20				
		50				
	58	20		2560 2554 2554 2554 2554	2550 2531 2531 2531 2531	2419 2455 2472 2489 2495
		50				
	59	20				
		50				
	0	20				
11		50		2554 2560	2478 2478 2525	2489 2495 2495
	1	20				
		50				
					2551	2484

The samples of pig iron were taken from the runner in a regular sample ladle and poured into the regular sample mold. The samples were all analyzed by Professor Demorest at Ohio State University, but only the total carbon was determined instead of combined carbon and graphitic carbon as was done in another test the following year. The analyses of the samples from cast No. 2457 were as follows:

ANALYSES OF SAMPLES FROM CAST No. 2457

Sample	Silicon, Per Cent.	Sulfur, Per Cent.	Total Carbon, Per Cent.	Temperature
No. 1	0.85	0.039	4.49	2450 (optical)
2	1.18	0.038	4.50	2538 (thermocouple)
3	0.80	0.039	4.41	2538 (thermocouple)
4	0.80	0.036	4.46	2556 (thermocouple)
5	0.75	0.034	4.44	2575 (thermocouple)
6	0.80	0.033	4.49	
7	0.80	0.029	4.52	

A cast that was quite regular in analysis and temperature was as follows:

ANALYSES OF SAMPLES FROM CAST No. 2465

Sample, No.	Silicon, Per Cent.	Sulfur, Per Cent.	Total Carbon, Per Cent.	Temperature, Deg. F.
1	0.99	0.026	4.54	2518 (optical)
2	0.99	0.027	4.57	2538 (thermocouple)
3	0.99	0.027	4.51	2541 (thermocouple)
4	1.03	0.023	4.57	2569 (thermocouple)
5	1.03	0.027	4.51	2543 (thermocouple)
6	1.03	0.026	4.63	2526 (thermocouple)
7	1.03	0.027	4.43	
8	0.94	0.027	4.51	

The highest silicon of any cast during the test was 1.17 per cent., and this is the cast that had the highest maximum temperature (2715° F.). The carbon was high with a maximum of 4.68 per cent., but there were two casts with higher carbon with 4.73 per cent. and 4.76 per cent. maximum carbon.

The temperatures of the slag in the two flushes preceding this cast were also rather high though not the highest in the test. The temperature of the first flush was from 2700° to 2758° F. for the last 5 min. of a flush that ran for 9 min. The second flush had temperatures from 2689° to 2715° for 5 min. with all above 2700° except the first and last minutes.

The iron at East furnace was running so regularly during the test that there was no chance to take the temperature of "cold iron." The nearest approach was cast No. 5769 at the West furnace, which showed the lowest total carbon and the highest sulfur. The low carbon gave the only clue to the carbon line shown in Figs. 1, 2 and 3. The record of cast No. 5769 is as follows:

ANALYSES OF SAMPLES FROM CAST No. 5769

Sample	Silicon, Per Cent.	Sulfur, Per Cent.	Total Carbon, Per Cent.	Temperature
No. 1W	0.66	0.051	4.04	2515
2W	0.66	0.049	4.05	2519
3W	0.66	0.042	4.12	2521
4W	0.66	0.044	4.11	2506
5W	0.66	0.049	4.13	2553

The slag temperatures were 2623° maximum and 2694° maximum, respectively, on the two flushes. Using the results from the one cast at West furnace and from all the casts for 5 days at the East furnace, three sets of curves (Figs. 1, 2 and 3) have been plotted showing these determinations.

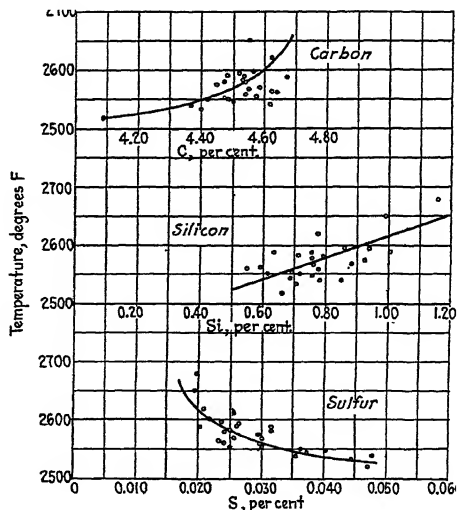


FIG. 1.—TEMPERATURE TESTS AT COLUMBUS BLAST FURNACES; AVERAGE TEMPERATURE VS. AVERAGE ANALYSES.

It is evident that, within the limits of this particular test while making basic iron with by-product coke, the carbon and silicon tend to increase with increased temperature, and that the sulfur decreases.

IRON AT THE TUYERES

Where does the carbon enter the iron, at the tuyeres, or below the tuyeres? From the few samples taken at the tuyeres during this test, we believe that the reactions that determine the character of the slag and

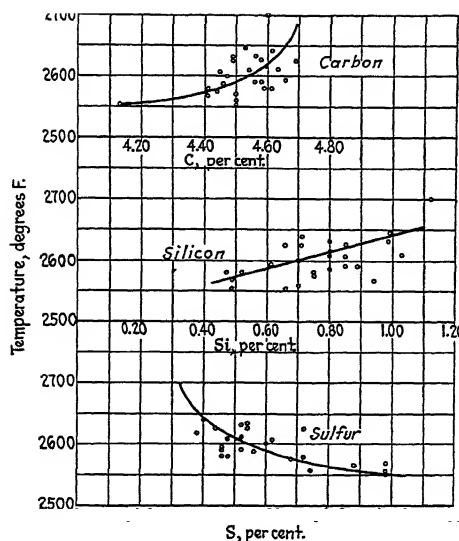


FIG. 2.—TEMPERATURE TESTS AT COLUMBUS BLAST FURNACES; MAXIMUM TEMPERATURE VS. ACTUAL ANALYSES.

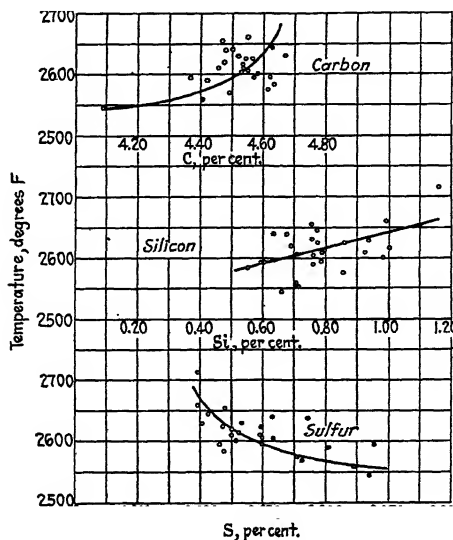


FIG. 3.—TEMPERATURE TESTS AT COLUMBUS BLAST FURNACES; MAXIMUM TEMPERATURE VS. AVERAGE ANALYSES.

of the iron are only partially completed at the tuyere level, and that much chemical action takes place between the level of the combustion zones in front of the tuyeres and the molten iron that collects in the bottom of the furnace. The results shown in Table 2 are indicative but not conclusive.

TABLE 2.—*Analyses of Metal and Slag Samples Taken at Tuyere Level When Gas Samples Were Taken*

Description	Silicon, Per Cent.	FeO, Per Cent.	Silica, Per Cent.	Phosphorus, Per Cent.	Manganese, Per Cent.	Alumina, Per Cent.	Lime, Per Cent.	Magnesia, Per Cent.	Total Carbon, Per Cent.	Sulfur, Per Cent.
Slag from end of sampler #3 Tuyere, East Furnace, May 4, 1923, 9:30 a. m.		13.29	31.60		0.70	16.71	36.25	3.41		0.250
Slag from #3 Tuyere, West Furnace, May 4, 1923, 1:30 p. m.		31.77	23.00		0.88	9.23	30.23	3.19		0.200
Metal from end of Sampler #3 Tuyere, East Furnace, May 4, 1923, 9:30 a. m.	2.82				0.34				1.55	0.095
Metal from #3 Tuyere, East Furnace, May 5, 1923, 9:30 a. m. #1 blast pres.	3.34			0.257	0.48				2.78	0.020
Metal from #3 Tuyere, West Furnace, May 4, 1923, 1:30 p. m.	0.66			0.297	0.37				2.23	0.101

DISCUSSION

J. T. MacKENZIE, Birmingham, Ala.—The total carbons in some instances look rather high to me. Were these poured very rapidly from the runner into the chill? It seems that they were far above the saturation point at the freezing temperature, but if you poured them immediately from the runner into the chill, you would catch more carbon in your pig iron.

R. H. SWEETSER.—These samples were taken right from the pig iron as the iron came into the dam and were poured into our ordinary cast-iron mold.

E. J. LOWRY, Chicago, Ill.—Then these total carbons are not what you ordinarily find in your run of pig iron. For instance, you have 4.94 per cent. in one instance and 4.96 per cent. in another. They are much higher than we are accustomed to run in the average periods of malleable.

G. DAVIES.—These carbons were determined in very low-silicon iron and neither of them would support Mr. Sweetser that the silicon content is not reflected in any way in the percentage of carbon.

R. MOLDENKE, Watchung, N. J.—How were the samples taken for making your determinations of combined carbon and graphite? I never believe that an analysis for graphite and combined carbon is authentic unless I know from what point of the casting it comes. When we took the whole of the chips entirely across a section we got a sort of average which was comparative with different irons. So, I would like to find out how the samples were taken, to see if your samples were taken so that you can compare one sample with the other.

R. H. SWEETSER.—Our observations so far show that the total carbon at a given furnace varied comparatively little.

You asked me where this test was taken, and I gave you the first analysis. Here is another where the runner showed 4.35 per cent. total carbon, 4.25 per cent. in the ladle

and 4.25 per cent. in the car, while the graphitic carbon and the combined carbon varied considerably; that is, in the car the combined carbon went up to 4.14 per cent. We are going to find in this investigation that the total carbon seems to be a characteristic of a particular furnace, for a particular fuel.

In my table, I have given the analysis of a pig iron which was made 29 years ago out of Broadtop coke. Mr. Ross, of Riddlesburg, has an analysis that was made last year with coke from the same region, and I would be glad if he would read that.

E. P. Ross, Riddlesburg, Pa.—The total carbon seems to vary very little in one particular furnace using one particular coke, but that does not indicate that other furnaces using different cokes will not bring out different total carbon contents.

We operate on a soft Broadtop coke and our total carbons run down to 3.50 and up to 3.80 per cent., but we observe that there is a tremendous oxidation going on after the iron has come from the furnace. It seems to be supersaturated with the metalloids that come out in the big ladle; there is tremendous oxidation going on there, and before we finish pouring the cast, the carbon may be very much lower than when it first came out of the furnace. There might be 6 or 8 in. of metalloids on top of a cast of iron before pouring into the pig mold had been finished. The silicon was 2.90; sulfur, 0.025; phosphorus, 0.59; manganese, 0.69; total carbon, 3.85; combined carbon, 0.23; graphitic carbon, 3.62, and copper, 0.076 per cent. This cast was made from an all-ore mixture.

We notice that even the surface of the iron, after it is chilled, is blistered; there is an oxidation going on continually, and there will be little bubbles coming up to the top of the iron producing a honeycomb effect through the top of the iron for which we cannot account. We thought perhaps it might be copper that was the cause of it, but frankly we do not know.

E. J. Lowry, Chicago, Ill.—I think Mr. Sweetser has brought out the point that in the lower ranges with silicon, such as No. 2 Foundry and No. 2X Foundry, no difference is found in the total carbon. That has not been the experience of most of the pig-iron sellers and consumers. For instance, if you go to a malleable foundry using a 1.75 per cent. silicon, you will always get a higher total carbon running in that iron than you will in a foundry running 2.75 per cent. silicon.

You will find there is a considerable amount of metalloids along with the slag, and the more the ladle stands, the more carbon occurs.

As to boiling in the mold and the layer of sponge, that, to my mind, has always been caused by the liming of the mold. If you lime with a straight calcium carbonate, the calcium carbonate breaks up with the CO_2 which in turn passes through the carbon. If you use a lime with a straight calcium carbonate, you do not get the evolution in the mold that you do when you coat it with a straight lime or a partially burned lime. I think one of the greatest tendencies in the foundry industry is to run into trouble when you lime the mold. It is a mechanical feature that enters into the casting of iron rather than in the smelting.

W. B. Coleman, Philadelphia, Pa.—I have seen analyzed thousands of samples of pig iron from practically all of the blast furnaces around the East, and since there is no phosphorus mentioned here, I assume that the foundry iron might contain phosphorus up to around 0.50 per cent. I do not recall ever seeing a total carbon over 4 per cent. on a foundry iron where the phosphorus is up around 0.50 per cent. About 3.85 or 3.90 per cent. might be the highest, while on malleable iron where the phosphorus is down around 0.15 per cent., I have seen the total carbon around 4.15 or 4.20. I have never handled an iron where the total carbon has been over 4.20. What is the phosphorus on that foundry iron?

R. H. SWEETSER.—It was made 29 years ago and I do not know.

W. B. COLEMAN.—I would like to know how today they can make a soft gray iron with large graphitic flakes, with about 4.25 per cent. silicon.

SECOND SESSION

PROCEDURE OF OPEN-HEARTH COMMITTEE IN STUDYING PIG IRON

C. H. HERTY, JR.,* Pittsburgh, Pa.—The procedure of the Open-hearth Committee in studying pig iron and its effects in the open hearth has been influenced by the discussions at a number of meetings of the Committee. Those discussions have brought out very clearly that, as far as the pig iron affects the quality, interference with quality should be studied for three definite types of pig iron: (1) low-silicon pig iron; (2) very high-silicon pig iron, and (3) pig iron in which the silicon varies over a wide range in short periods, such as 1 to 3 casts.

If a blast furnace is running steadily at 1 per cent. silicon iron, practically no trouble, apparently, can be traced to the pig iron, but if the furnace is running from 0.60 up to 1.50 and then down to 0.60 per cent., trouble in the finished steel is liable to occur, which apparently cannot be corrected by open-hearth practice.

The open-hearth discussions have shown fairly definitely that with low-silicon iron, one type of trouble arises on the finished product and with the high-silicon iron, another. Our experience with the high-silicon is that any non-metallic matter that may form, does so when the pig iron is put into the furnace; that is, silicates immediately form which are more liable to be high in silica than would happen with low-silicon iron.

I think Mr. Sweetser will agree with me that if a blast furnace is very irregular, different points in the furnace may be working irregularly, in other words, the temperature gradient all over the furnace is not uniform. We have made many determinations on that point which indicate that a varying blast furnace causes hot spots and cold spots around the periphery. If some of those spots are hot and others are cold, although the analysis of the iron may be constant, then unreduced oxide, going through on one side, and good iron, going through on the other, may mix and the average temperature of the iron is not high enough to get completely rid of the oxides. So our method of attack is directly on that point; it is our first objective.

Our procedure is to make a number of consecutive tests at a blast furnace, making a detailed study of the blast-furnace operation. We follow the iron through the open-hearth heat, being sure that we know just how much of each kind of iron gets into the open hearth, and take out specified samples at certain times during the process, working on steel where no finishing additions are made in the furnace. We are following one type of iron through the billet mills and into the finished product and are obtaining complete data on open-hearth operation.

Besides the ordinary chemical analyses on the samples of slag and metal, we are running samples of the pig iron, the steel in the open hearth and the finished steel, for oxygen and silicates. We are going to find out whether or not oxygen, as silicates or oxides, has some connection with this trouble. If not, we may be able to locate some other factor. I am inclined to think that the silicates are responsible, judging from the indications in some other work that the Bureau of Mines is doing in Pittsburgh.

We have laid out a very definite method of testing to which we will adhere at the different plants where we are now working and at others in the future. Our aim is to

*Mr. Herty presented for the Open-hearth Committee a report upon its procedure in studying pig iron and its effect in the open hearth.

obtain all the data from the different plants so as to compare them accurately without estimation or guesswork. We want to get all data quantitatively as accurately as possible, which, when correlated, will show what effect pig iron has on open-hearth steel.

The form of the carbon in the pig iron probably has no direct bearing on the open-hearth process, or the quality of the finished product. On the other hand, we do not know whether it is an indication of some other condition.

From the discussion it seems to me that if the foundrymen and blast-furnace men are going to cooperate in research, they must work from fundamentals. Whenever a man says, "I have tried everything," I know that he has perhaps tried 5 per cent., and, furthermore, that he has "tried everything" with all the variables present. I do not think any progress will be made if they try to pick out all the variables at the same time and put a finger on the trouble.

R. H. SWEETSER.—When we attempt work like this, we always begin with some preconceived notions, some of which on the present subject have already been changed, for example, that carbon might be the cause of the trouble.

When J. E. Johnson presented at Cleveland his paper on "Blast Furnace Slags," two different schools of blast-furnace men came there with certain preconceived notions, and we found that all of us were wrong. The problem was one of temperatures, and at the time pyrometers were not made that would give the temperatures. Incidentally, the slag work that has been done since then has completely upset our preconceived notions.

L. F. REINARTZ, Middletown, O.—The blast-furnace men as a whole have not considered quality as much as some other points. They think primarily of tonnage, then of cost, of fuel consumption, and lastly probably of analysis. The physical constitution is more important than all the other factors combined. In the manufacture of any grade of open-hearth steel, the analysis means less than 50 per cent. An open-hearth heat may have a perfect analysis and yet not be worth scrap; and one that is slightly off analysis may make a very good heat, simply because the physical conditions are right. I believe, as time goes on, we are going to stress more and more the physical rather than the chemical end of the blast-furnace work.

Mr. Coleman said he had never observed an instance where pig iron had caused trouble in armor plate or in shell steel. Our experience has been similar. In the higher-carbon steel not nearly so much trouble is experienced as in those lower in carbon, particularly in those carbon steels that are killed with silicon. In such a heat a heavy slag is carried; before tapping, it is deoxidized as thoroughly as possible, then silicon, manganese, titanium, aluminum and whatever else is desired, is added in the ladle, and in that way any oxidizing effect that might have been caused by the pig iron is masked.

It was also stated that an open hearth is an oxidation process. We know that to be true, but if it is so, there must be oxidation control. There is a certain kind of pig iron made in a blast furnace which will cause very serious trouble in rimming steel—say 0.10 per cent. carbon for sheet purposes. Such pig iron will cause a bath to melt cold, the slag will be dead, will not be active, the carbon will drop very rapidly, and when the metal is poured it will contain much gas; when it is rolled into sheet bars or into sheets, it will have many laminations and blisters. If you put the silicon into the ladle, you do not have these losses due to blisters.

In our practice we use cold pig and, therefore, we can check a little bit closer than when hot metal is used, because occasionally we can break a piece of the metal and examine it. We sometimes find that the grain is not uniform; we also have a blowhole condition on top of the pig iron, which was previously explained at this meeting.

When the pig iron is right, we do not have this appearance. It may be due to the lime wash used.

We have also discovered in our own practice that, when the manganese is around 1.50 to 2 per cent., our blast-furnace product is more regular. Our contention has always been, with Mr. Sweetser, that we must have high total carbon. The question of combined and graphitic carbon is very hard to solve. You must know exactly the conditions under which the metal was chilled, in order to compare graphitic and combined carbon results. We have made all kinds of analyses and find out they do not mean very much. You can anneal the pig iron and change the graphitic and combined carbon content at will. It is the total carbon that counts.

We want total carbon high in the pig iron because, if the carbon is high, we know we use less pig iron per ton of steel made, and scrap is cheaper ordinarily than pig iron. From past experience, the pig iron that gives most trouble is generally under 0.75 per cent. silicon; when the silicon is up around 1 per cent. and is coming from a physically hot furnace, in our own practice, we get the very best results.

When a blast-furnace man is trying to lower costs, he sometimes goes to a considerable length to get 50 lb. lower coke per ton in a month's time. That probably will save 10 to 15 c. per ton. If he loses 0.50 per cent. of the open-hearth product at the end of the bar mill he loses that 10 c. on account of pipe or blisters if the pig iron is bad.

We have had very good results with some pig iron as low as 0.60 per cent. silicon, with 0.020 per cent. sulfur. We cut the limestone percentage and thus had less slag volume, but if we had 0.60 per cent. silicon and 0.05 per cent. sulfur we encountered considerable difficulty.

C. D. KING, New York, N. Y.—Have any of the blast-furnace men here been using open-hearth slag as part of their charge and, if so, have they noticed any perceptible difference in the total carbon content of the pig iron so produced, as compared with the basic pig iron ordinarily produced?

H. J. ALLEN, Cleveland, O.—We use a small quantity of open-hearth slag at various times. Personally, if I can get it, I prefer not to use slag. It is probably the most variable element we put into our furnace.

E. P. ROSS, Riddlesburg, Pa.—At our plant we have used about 3 to 5 per cent. slag for manganese content; we carried it to get the manganese out of it. We never made any carbon determinations or observed any other effects. The open-hearth slag used carried about 10 per cent. manganese. When our mixture was rich, we carried it to get a larger slag volume.

C. D. KING.—I note that the open-hearth men prefer the high-carbon content. Would a higher carbon content indicate a more stable pig iron?

L. F. REINARTZ, Middletown, O.—Yes.

C. D. KING.—How do you reconcile that with some foreign processes now being used, the carbon being brought down before it goes into the open hearth?

C. H. HERTY, JR.—That is all right provided your carbon is high when you start.

C. D. KING.—Does that mean high-carbon pig iron, 5 per cent. carbon, 50 per cent. charge, is preferable to a $3\frac{1}{2}$ per cent. carbon, 50 per cent. pig-iron charge?

A. L. FEILD, Canton, O.—The silicon and manganese have some effect in connection with the carbon.

R. H. SWEETSER.—Mr. King has brought up the same problem with open-hearth pig iron that Dr. Moldenke brought up in regard to foundry iron, and the blast-furnace man does not know whether certain analyses are indicative of the right kind of iron or whether the elements themselves make that right kind of iron. Those of

us who made pig iron that was sold on fracture know that there is a difference between 1X and 2X iron. The analysis that Mr. Ross gave is of a pig iron that had the combined, the graphitic, and the total carbon; it was a 1X iron. A 2X iron made in the same region, from the same kind of coke, had 3.14 per cent. graphitic carbon, 0.74 per cent. combined carbon, and 3.88 per cent. total carbon and 2.79 per cent. silicon. They were taken 29 years apart. We cannot testify but we will agree that they were both commercially correct. They were both graded irons and sold on the grade.

The point I want to make is that the 1X iron had nearly all of its carbon as graphite, all excepting 0.23 per cent.; the 2X iron had all of its carbon as graphite excepting 0.74 per cent. That difference between 0.23 and 0.74 per cent., according to the analysis which was good practice at that time, will show the difference between the two structures of those two pig irons. What we want to find out in our research work is whether these analyses are going to tell the whole story. We do not know.

R. MOLDENKE, Watchung, N. J.—I think there is one more factor you must have there to compare those two analyses. If the 1X was drilled straight through so that you got all the borings from one hole, and if the 2X was drilled three or four times to small depths, you would not be able to compare those two samples at all. So you must know more about your sample before you can judge combined carbon graphite. Total carbon is all right.

E. J. LOWRY, Chicago, Ill.—You would have to get the different factors concerning the amount of carbon in the different pigs.

L. F. REINARTZ.—I know that the difference between the combined and graphitic carbon is large. Recently we took quite a number of different pigs from different heats, from different blast-furnace casts, and we analyzed them as they came to us. We drilled the pigs all the way through and found that the combined carbon varied about 0.60 to 1.25 per cent. The 1.25 per cent. cast happened to be about 0.50 per cent. silicon, but results varied widely. Then we heated those pigs in an electric furnace to about 1300°. We then drilled and analyzed them again. All, with the exception of the one that was very high—the 1.25 per cent. cast—dropped uniformly to about 0.20 per cent. We cooled them off and heated them up to the same temperature and found the 1.25 per cent. showed almost the same percentage. We heated them up twice more. The very low silicon cast came down to 0.50 per cent. and stayed there. The rest remained about 0.15 or 0.20 per cent. combined carbon. Such a test may be one way to determine the ultimate carbon combined in the pig iron, because we showed in a good many tests that the regular pig iron came down to the same percentage each time.

D. J. DEMOREST, Columbus, O.—Since we met, I have been entertaining doubt as to the role that oxygen plays in the pig iron. I can understand, possibly, that there is another factor which is entering in and which is serious. If it should happen that a particular cast of pig iron is high in colloids—it is an emulsion—then it can also be imagined that when it is remelted, it will still stay emulsified or colloidal in nature, unless something is added to act as a flocculating agent.

Mr. Reinartz referred to the fact that high silicon in the melt in the furnace usually resulted in a satisfactory steel. I can imagine that the silicon, on being oxidized, acts as a flocculating or a cleansing agent to take out this colloiddally-suspended material.

Dr. Moldenke spoke of his experience and belief that high-carbon pig is safer in the foundry than the low-carbon pig. Now, it is readily conceivable that when a high-carbon pig cools and precipitates out such a large amount of graphite, that graphite will act as a cleansing agent to clean out the suspended colloidal material. An enormous volume of that carbon is precipitated, almost as great a volume as that of

the iron itself. If you get the colloidal material out of the iron and into the graphite only you have accomplished the decolloidalizing effect.

During the early stages of the war, I examined many different pieces of steel that failed in mechanical tests. They were shell steels that were all right chemically and microscopically, until a good-sized specimen was polished highly and put under the right light. Then you could tell by the naked eye that the steel was decidedly non-uniform. Considerable sections of the surface gave a much duller reflection than the rest. Obviously a colloidal suspension had occurred, the colloids being too small to see with the microscope, but together they gave a different effect as compared to the parts that did not have colloidal suspensions.

A. L. FEILD.—We may be paying too much attention to the properties and structures of the pig iron. The open-hearth man is interested only in one condition; that is, the carbon at which his bath melts, and how much manganese and silicon are in the bath at the time of melting. In other words, steel is made from molten cast iron, in any event. The melt-down is a process where considerable oxidation of scrap occurs, and the manganese and silicon specifications in the pig iron simply determine how much total manganese and how much total silicon will be in the slag. The one variable, though, is the carbon at the melt-down; that depends largely on the type of scrap used. Unless you can get exactly the same scrap on two consecutive heats, it is almost impossible to duplicate the conditions at the melt-down. I do not see just how we can control these conditions accurately.

R. MOLDENKE.—What is this oxidized product? That ought to be better understood. We in the foundry believe that the question of oxidation might be twofold. For instance, a rusty piece of iron will not affect at all the quality of the metal that goes through; the rust goes into the slag. However, an oxide of iron dissolved in the iron, goes through into the resulting product. So we ought to have in mind that the oxygen we speak about is not oxygen per se but is FeO dissolved in the iron, which the iron is actually capable of dissolving. Very likely, when at a high enough temperature, it reacts. For instance, reference was made to the effect of the carbon in the open-hearth process, which is an oxidizing process, but towards the end the iron is deoxidized. We cannot do that in the foundry. I have seen it done in the foundry only once with manganese when they used a lot of steel in the mixture and the temperature of the low-carbon material was much above what it ordinarily is; then it did deoxidize the iron. But under normal conditions you do not get that deoxidizing process. If I added 0.10 per cent. titanium, it disappeared, went to slag, but it deoxidized.

Remember that we would like to have much more data about the oxygen in that iron, which it is oxidizing. The main problem is to get it out. For that we need high temperatures and deoxidizing materials which we do not have in ordinary foundry practice.

C. H. HERTY, JR.—Professor Demorest, just what do you think this colloidal matter is, if it is not oxide?

D. J. DEMOREST.—Of course, it is oxidized material; that is, it is silicon, manganese, iron, aluminum or calcium oxides, and the suspended material may be any of the oxides.

C. H. HERTY, JR.—That is just the line we are following.

D. J. DEMOREST.—The point I want to make is that the presence of these suspended oxides in the pig iron does not necessarily mean any different condition of oxidation in the blast furnace. It may mean something else that has caused a more intimate suspension of this material in the pig iron.

A. L. FEILD.—Dr. Moldenke spoke of deoxidizing the pig iron. The two deoxidizers that are most widely used in steel are silicon and manganese. Manganese is probably the chief deoxidizer, if by that you mean actual removal of oxygen. Silicon probably fixes the gases so that the steel remains solid instead of full of blowholes. But cast iron has a great deal more silicon in it than any steel, except a few special steels; it has as much and usually more manganese than the ordinary steel. The presence of high carbon should help deoxidation rather than the reverse.

So it seems to me that the presence of dissolved oxygen in cast iron is almost impossible. It must be a suspension or an emulsion of silicates. I do not see why, if you can kill a steel with 20 points of oxygen, you cannot deoxidize it with 2 per cent.

R. MOLDENKE.—The temperature in the case of steel is 300° or 400° higher than for cast iron.

A. L. FEILD.—Suppose you heat the cast iron hot enough.

R. MOLDENKE.—I think that is the very basis of these new cast irons that are at such high strength. Imagine 3000° F.!

J. S. VANICK, Bayonne, N. J.—There is a big difference between an oxide formed in rusting and one that is formed in the grate-bar surface. A small amount of grate-bar scrap might be enough to account for the oxide present as silicates.

E. J. LOWRY, Chicago, Ill.—Answering Mr. Feild, both Mr. Piwowsky and Mr. Thyssen point out that the heat does not change the composition of the metal.

A. L. FEILD.—Mr. Piwowsky has merely succeeded in holding the cast iron at high temperatures without any change in composition but with changes in the properties of the cast product, which may mean that hot metal can change some, depending on how hot it is in the blast furnace.

M. GRINDAL, BERGEN, Norway.—I have had some experience which supports very strongly the views of Dr. Moldenke. I worked with the electric furnace on the west coast of Norway, where I tried melting cast irons for various purposes. We tried pure steel, making it synthetically, and got cast iron of a tensile strength of about 26 tons per sq. in. I also experimented with very rusty scrap and at first had very poor results, in spite of the fact that the top temperature reached during the mill was so white hot that it was practically the color of molten steel. However, that was in a very rapidly heating furnace. If I kept the material there for some time, I got a very good product which I would consider to be deoxidized. No doubt at those high temperatures, when practically duplicating the conditions in a steel furnace, gray cast iron is deoxidized, and the tendency in your country now is all along the line of melting hotter in the iron foundry.

In the cupola, the process was developed in the Thyssen works by Dr. Engle; they are using up to 80 per cent. steel. It also works on the principle of very hot iron, using very careful charging methods, and a furnace has been developed which is a combination of an air furnace and a cupola, bringing the temperature up to about 3000° F. Those methods in all probability would be of some benefit if duplicated in this country.

R. MOLDENKE.—In this little air furnace, with the shaft of the cupola, Dr. Thyssen charges as high as 96 per cent. steel. As high as 90 or 95 per cent. steel can be put up, carefully laid, with enough coke, preferably petroleum coke, in between these charges, and ferrosilicon and ferromanganese added in the shape of the briquets that they make in Europe. That makes a total carbon of about 2.50 per cent., silicon very near 3 per cent., and cast iron, 60,000 lb. tensile, or very nearly. But, as Mr. Grindal says, it must have time to get the reaction between the furnace and absorb the

carbon. An air furnace of continuous operation is made by this process. It can run all day long. It is the new development, but that has nothing to do with the fact of making bad pig iron.

J. T. MacKENZIE Birmingham, Ala.—On the question of oxidation and high temperature, it is very easy to confuse two conflicting influences. Dr. Piwowsky, in his high-temperature work, showed conclusively, I think, that the reason for the great strength of high-temperature irons lies in the absence of graphite nuclei. That makes a very easily machinable casting, but you can also deoxidize in the same furnace and at the same time, and if you get good, machinable iron, you will be deoxidizing it.

However, all the troubles of the foundryman will not disappear when he gets his iron deoxidized at high temperature. Elliott at Cincinnati and I at Birmingham experimented with Duplex gray iron for a long time. The deoxidation of the iron in the electric furnace does not add fluidity and it requires about 0.50 per cent. more silicon to produce an iron of the same fracture and Brinell hardness. Probably that effect is due primarily to the elimination of graphite nuclei.

In proof of the influence of the graphite nuclei, I had occasion at the Bureau of Standards to get some of those little buttons of molten steel which result from the melting down of the oxygen determinations; a small sample of some 25 gm. is melted in an induction furnace in a graphite crucible and held at 1450° C. for 1 hr. under vacuum, causing absorption of carbon up to approximately 5.25 per cent. The silicon is practically nil. Dr. Burgess was investigating rivet steels for the A. S. T. M., the steel was pure.

When you cool the steel by cutting off the current, it drops from 1450° C. to black heat inside of 10 min., which certainly would not be called slow cooling, all but about 0.8 per cent. of the button is graphite, great flakes of primary graphite interspersed with fine flakes of secondary graphite, and the matrix is pearlitic, in fact, you have nothing but pearlite and graphite, which upset all of my ideas about graphitization.

I got Mr. Jordan to hold a sample for me at 1130° and I found only 4.30 per cent. total carbon in the melt; in other words, all the excess graphite came to the top as kish and you could see it accumulated as a spongy mass on top of the button, but the graphitization was practically complete to the same point. It cooled rapidly from 1130° and the combined carbon in the sample was only 0.8 per cent., so that while the iron had really freed itself of most of the graphite nuclei, there was still enough left to initiate the graphitization and it proceeded practically to completion.

D. J. DEMOREST.—Holding the metal at a high temperature not only induces reduction of oxides by carbon, silicon and manganese, but it also decreases the viscosity of the iron and gives it a chance to become de-emulsified.

C. D. KING.—In regard to this high-carbon pig iron which is preferred by several of the men here, they prefer it, as I understand it, because it is indicative of a more stable pig iron. Now, is it possible to produce this high manganese iron with a lower carbon content? It is hardly necessary to point out that in the ordinary charging of the open-hearth furnaces, apart from the questions of scrap and pig iron supply, which of course will decide the amount of pig iron charged in many cases, there are times when the physical conditions of the plant will necessitate a higher charge of pig iron to facilitate the charging. If that is going to be discounted by a higher carbon pig iron, you are going to lose the advantage of the higher pig charging in addition to the loss of time from the higher carbon content and loss of additional metalloids.

E. J. LOWRY.—There is one feature that the blast-furnace men might consider, that would help the foundrymen without perhaps hurting the blast-furnace trade, and that is the size of the pigs. We have found in extended research that the smaller

the pig the less the loss and trouble occurring in the melting practice. We found that 40-lb. pigs gave us a hotter temperature with less coke than when using the 105-lb. pig. At the same time we found that where the 40-lb. pig was cast in thin slabs, the iron set so quickly that it could not boil in the mold and practically all that scruff that was on the top was eliminated.

I know there is a limit to which you can go in order to suit the capacity of the machine, but it would seem that where a single-strand machine is in operation if they should go to a double-strand machine and cut the pigs in two, the foundry industry would be greatly aided.

R. H. SWEETSER.—Mr. Lowry, are you not increasing your combined carbon when you have small, flat, chilled pigs?

E. J. LOWRY.—I think that is exactly what you want to do as far as the foundry is concerned, because the higher the combined carbon in the pig iron and the higher the total carbon, the lower the temperature at which it will melt. The lowest temperature melting material will come down hottest at the spout, and you want hot metal.

R. MOLDENKE.—I made the first determinations of the melting point of cast iron way back in my early career and we found that the melting point depended upon the combined carbon altogether. In Europe you will find that their casting machines make pigs exactly the size of the sand-cast pig. In his developments, Piwowsky has shown that he not only wants a machined pig cast thin, but he actually wants cold water thrown on it so that he gets all of the graphite nuclei wiped out.

E. P. ROSS.—We have cut the size of our pig about 50 per cent., for probably two reasons: (1) We felt that our customers were complaining about the weight in handling; we cater to the smaller foundries. (2) We felt that the small pig gave a larger surface and was easily melted and accomplished what the previous speaker claimed, but we found, on the other hand, that when we got down to those smaller sections the pig did not grain up as well. Then some of our customers came back and wanted the fracture pig and we got into a little trouble in that connection. We found by not chilling the pig and giving it a chance to cool as slowly as possible, that the fracture would come back to it. So we have been working along the line suggested.

K. C. MCCUTCHEON, Ashland, Ky.—This discussion certainly shows the need for intelligent research. For instance, Mr. Ross mentioned that they tried to make it more satisfactory for the foundries by making a smaller section and then the foundries wanted the smaller section and the fracture they were accustomed to. In other words, they wanted their cake and they wanted to eat it too.

In our research we must look for the fundamental features with an open mind, not on the basis of our past experience but with a view to the results that we are going to get. The Open-hearth Committee is going at it with that in view. In fact, both the open-hearth and the blast-furnace departments are creating conditions and observing the results with the idea of determining something, positively or negatively.

It seems to me that it might be well for the open-hearth man who is melting cold pig iron to consider the small pigs also, because he is interested in melting the pig iron fast because he knows he saves oxidation. However, that is a mechanical feature that could be worked out between the open-hearth man and the blast-furnace man, quite aside from the scientific line that we are bound to follow in examinations.

We can investigate, helped by our past experience, but let us not try to judge by our past experience; in other words, our discussions show, I think, that the open-hearth men withhold a little, the blast-furnace men withhold a little, and the foundrymen withhold a little also, because they are speaking to a mixed gathering and do not want

to expose spots supposedly vulnerable. We find out, as we investigate intelligently, whether or not to hold these old bugaboos. I know that the Open-hearth Committee is open-minded as to the process they are going to use. So far they have decided that they are going to follow a certain line of investigation but they are perfectly willing to accept suggestions from any source and they have been getting a lot of "ammunition" here today in the discussions that we have been having.

J. T. MACKENZIE.—I agree with Mr. Lowry about smaller-sized pig iron, but I do not agree with him that the higher the combined carbon, the hotter it will melt. If it starts down through the superheating zone at 1300°, it will come out hotter than if it began its journey through that zone at 1150°.

R. MOLDENKE.—I agree with Mr. Lowry and disagree with Mr. MacKenzie. In my early days I made a lot of experiments with Thomas D. West. Finally, we made some of the same irons poured into chills and poured into sand. The discussion was: Which melts first? (It went through all the papers at the time.) Then I stuck a 3-ft. tile right into the furnace and put these two pieces on top of it. I observed that the chill pig kept its shape throughout the whole melt but ran down in drops. On the other hand, I found that gray pig kept its shape for a while until it became hotter and hotter, and then all of a sudden just sort of collapsed. I took a 1¼-in. bar and touched these two pigs and I found that the white one was hard and the gray one squashed out. So you see in gray cast iron there is a stage between solid and liquid which makes it mushy because the graphite dissolved in the solid state.

When a drop of iron forms on melting pig iron or scrap in the cupola, the moment it drops off, it is not going to get more heat, no matter how it goes through. In other words, you must give the full superheat at the place where it drops off the pig. At least, that is my experience. You can not give it heat by having it drop in further through the cupola because it goes too fast. So you must get the heat in right away, and the smaller the pig is, the quicker the heat will go through.

J. T. MACKENZIE.—If you put in a material melting at 1150°, it will begin its rapid journey down at 1150°. That point will be above the melting zone. If you put in material which does not melt until 1300°, it does not begin its descent until it has already attained its temperature of 1300°, and as both must pass through the same distance exposed to the maximum temperature of 1700° or 1800°, you must have more heat in the second case.

E. P. ROSS.—We have two questions here—two points. One is physical and the other chemical. One is a question of how much carbon melts these pigs. It is evident that, other things being equal, the larger the surface exposed to the heat, the quicker the iron will melt and go into the hearth. To my mind, it is better to have a small pig.

Now we can discuss which of the carbons melts easily, assuming that we go back to the small pig. To my mind we have decided on the small pig, due to the fact that it exposes a large surface and will absorb heat more quickly and therefore melt quickly.

A. L. FIELD.—There are two things we ought to keep in mind in this discussion. One is that a metal cannot take up more than a certain amount of heat up to the melting point, and that is always a definite amount; that is, no matter how fast it gets the heat or how fast it moves, the total heat at its melting point is always the same. If a particle of iron or a drop ran into the crucible, as soon as it was melted, without absorbing any heat, an increase in the amount of coke would not be able to increase the temperature of the metal in the hearth. If it is true that an increase in coke does increase the temperature of the metal in the hearth, it must absorb heat during its flow to the crucible after melting.

L. BRADLEY.—I would like to discuss this question of whether the material absorbs any heat below the tuyeres. Professor Richards' metallurgical calculations have shown in a number of instances that something like 20 per cent. of the coke is oxidized below the tuyeres. That is a very 'strong endothermic reaction; you must supply the heat for its reaction to take place. Therefore, I contend that there must be some heat given to the material below the tuyeres. One of the best ways to transmit heat from one body to another is by radiation, and surely there is sufficient temperature there to cause radiation below, as well as up.

R. H. SWEETSER.—Are you speaking of the tuyères of the cupola or of the blast furnace?

L. BRADLEY.—Of the blast furnace, but the principle is the same in both.

R. H. SWEETSER.—That was brought out in the last page of my own paper: that the pig iron had not received all of its carbon when it passed the tuyeres; in fact, it had received only about 2.82 per cent. and it went up to about 4.50 per cent.

L. BRADLEY.—A great reduction takes place below the tuyeres, as well as above.

T. L. JOSEPH, Minneapolis, Minn.—From blast-furnace experience, it is generally known that slags which have higher melting temperatures will produce higher temperatures in the hearth. Charcoal furnaces with low-melting slag run at low temperatures. I think the same principle may apply somewhat to a cupola.

A. L. FEILD.—Except that the cupola slag is not sufficient (is it?) to really preheat the iron.

T. L. JOSEPH.—The slag may be negligible but steel scrap that melts at a higher temperature will remain a longer time in the melting zone than cast iron and will absorb more heat before it drops out of the zone where it can absorb heat.

M. GRINDAL.—The knowledge that I have gathered about cupola practice is entirely due to American investigations. I do not think there is any country where the study and investigative work on cupola practice has been so exhaustive, so methodical and so thorough as in the United States. This work was originated by Dr. Moldenke, then taken up by the Bureau of Mines and further conducted there. The most recent work of American practice which I have studied was at the University of Michigan where observation points were made through various heights of the cupola, right up to the charging door, and it is conclusively shown that the point where a piece of metal melts in the cupola is practically independent of the chemical analysis, being solely dependent on the volume as compared with the surface of the piece in question. That is to say, if you have a small, thin piece of steel scrap with a very high melting point, it will melt higher up in the cupola than a large block of pig iron.

The tendency in Europe is to demand smaller blocks of pig iron, as mentioned here today, because we feel that we need to use less fuel, it is easier to handle and we get hotter metal. You can take a comparison from molding sands. If you want a very refractory sand, you use a coarse-grained sand. A very fine-grained sand will not stand up under such a condition.

The investigations at the University of Michigan showed that the cupola actually makes three charges at one time. The thin metal begins higher, and when the charge has passed the height of three charges down, the biggest lump of pig iron disappears. Now it stands to reason that to melt higher up it must have a bigger zone to travel and although it travels fast, it is bound to absorb more heat. Consequently,

we know if we take very thin scrap, we are bound to get a very hot iron if we use the same amount of coke as we do for heavy pieces of scrap, and I think that practical experience should support the demand for having smaller pieces of pig iron.

B. F. SHEPHERD, Phillipsburg, N. J.—Mr. MacKenzie, did you determine the combined carbon on that iron that was held at 1450° C.?

J. T. MacKENZIE: It was 0.80 per cent.

SATURATION POINT OF GRAPHITE

H. T. MORRIS, Bethlehem, Pa.—I did not intend to say anything from the standpoint of extensive knowledge of pig iron, which I do not claim, but I heard a gentleman say at this meeting that he always understood that the saturation point for carbon in pig iron was 4.20 per cent., and he said that in discussing some figures that Mr Sweetser had which showed 4.40 per cent. total carbon in pig iron.

It occurred to me that there is nothing essentially unbelievable about Mr. Sweetser's figure of 4.40 per cent., because the amount of carbon that you can find in a composition of carbon and iron depends largely on the purity of the alloy. If you are dealing with the binary compound, iron and carbon, you might have as much as 6.67 per cent. carbon in the iron as a clear compound. If you associate the iron and carbon with other materials, you have, of course, a limiting factor which prevents you from getting as much as 6.67 per cent., but we all know that when we load up iron with as much as 80 per cent. manganese, we can get somewhere between 6 and 6.50 per cent. carbon content. There, of course, is another factor contributing to that, namely, the hearth temperature of the furnace that made the material.

However, I think we make a mistake in discrediting figures that we read or see, because we have a preconceived notion that the figures are impossible. I do not think that 4.40 total carbon in pig iron is an impossibility. I do not think that figures a good deal higher than that are necessarily an impossibility. Most of us who have seen those figures are entirely unfamiliar with the circumstances surrounding the production of that pig iron and with the other constituents of the pig iron.

That brings me to another comment in the matter of research; there is a tendency always when an industrial firm or a private individual conducts a research, to discredit the figures if they do not agree with a preconceived notion. That has made me think sometimes that the only place to make research is in the Bureau of Mines or the Bureau of Standards or in some college where a research department is under the leadership of accredited people.

And then, being a steel man myself, I come to the next question, which has been discussed a number of times today, as to how to obtain your drillings from pig iron. It becomes quite evident, after slight consideration, that you cannot compare the drillings from different pieces of pig iron in the same way that you can compare the drillings from a ½-in. round or a 1-in. round bar of steel, which are the usual sizes adopted for specimens for comparisons. We know that if we take a small drill and drill the end of the bar halfway between the center of the bar and the edge, and if we take those bars from a certain location in each ingot, when we determine the drillings from those bars, we get relatively comparative results as between heats. But we do not get any such comparative results by taking drillings from castings or pieces of pig iron. We have a very different set of circumstances to deal with in taking samples from pig iron.

R. MOLDENKE.—I have gotten as high as 4.50 per cent. total carbon in my malleable practice when I had the silicon down to 0.25 and the phosphorus 0.20 per cent. But I think we who have seen thousands of analyses, when we find 4.85, draw a line through it and take it as 3.85.

J. T. MACKENZIE.—The iron-carbon diagram has been worked out very accurately by the most eminent scientists and we do know that the carbon absorption goes up with the temperature. I might say that the Bureau of Standards' buttons lie exactly on the iron-carbon diagram for the temperature of 1450°, but with commercially cast pig iron your samples were quenched, you might say, from the maximum temperature at which they could readily hold that much carbon, but the pig iron has been cooled considerably since then and anybody who has worked around a blast furnace knows that there is a continuous loss of carbon in those pig irons. I was simply questioning the authenticity of the carbons as applied to the product, not to your sample.

C. H. HERTY, JR.—I would like to comment on one point that Mr. Morris brought out about an individual company leaving research to the government bureaus and cooperative agencies. I cannot help but think that that is absolutely the wrong attitude to take. If individual companies had not taken up the research work, where would the General Electric and the Du Pont and the rest of those companies be?

E. J. LOWRY.—According to what Mr. Morris said, there is little doubt that you could get more carbon in a pig iron or in any iron sample than 4.20 per cent., if you put something in there to increase the solubility of carbon in iron, such as manganese. The same holds true of the Bethlehem Steel product "Mayari." You put chrome in that and it increases the solubility of the product.

Now as to this question of research along the line of fundamentals: Supposing you do work along the line of fundamentals and the fundamentals are uneconomic. Supposing you find that you should make all this pig iron out of virgin ore. You know yourself that you have a certain number of off-casts that are cheaper to resell at off-price. You are bound to sell them at a sacrifice. Take your tap cinder and flue dust: all that is considered scrap, but you have to put it in to get a certain product. Supposing you do eliminate them from the blast furnace, what are you going to use them for? It would not be economical, just because somebody says that 10 per cent. of the metallic content hurts pig iron, to throw it out, would it? You have to have a place to use them. The open-hearth people found that they could not use borings economically and they threw them out. Now you have found that, economically, you can use those borings.

It occurs to me that not only must the fundamentals be looked after but also the question of economics in the foundry and in the blast furnace.

C. H. HERTY, JR.—Besides carbon you also have silicon, phosphorus, manganese and other things bringing up the temperature tremendously. You have no idea of the effect of the silicon, the phosphorus and the manganese on the efficient manufacture of pig iron, and you cannot do any research work without taking all those factors into account in working out this problem.

E. J. LOWRY.—Mr. Sweetser, I do not think any one is doubting the accuracy of the analyses. I do not think there would be a question about these figures if all the tests were taken immediately from the tap-spout.

G. DAVIES.—Approximately what percentages of phosphorus and manganese were in your high-carbon tests, Mr. Sweetser?

R. H. SWEETSER.—It was all under 0.20 per cent. phosphorus and about 1 per cent. manganese at that time.

R. D. CARVER, Boston, Mass.—I was very much interested in these temperature determinations and I would like to know the details of this thermocouple. I am interested because the Thompson Laboratory of the General Electric Co. have developed a thermocouple which has stood temperatures of 2,500°. This thermocouple has been heated up some 100 to 125 times and checked each time, and we are trying to develop it for temperature determinations in the iron and steel foundries. At present they are using it on the Hughes quartz job. In a certain resistance furnace they have to keep the furnace within $\pm 5^\circ$ C. of 1,500 to 1,700° C.

D. J. DEMOREST.—It was a platinum-rhodium thermocouple, put inside of a quartz tube, and that in turn was put in a graphite tube. Because of the fact that the quartz tube becomes devitrified, it is only used once and then has to be thrown away.

R. H. SWEETSER.—We burned up several of those pyrometers. Of course, we had to pioneer on the whole thing. We had Professor Demorest and a group of students from the University make observations. We had to make observations every 30 sec., so we had them there day and night. At the same time the Bureau of Mines was making a series of investigations. Also, we took the temperature of the slag and the analysis of the slag. We literally had thousands of determinations and observations, and we are delighted to hear that they have found something that will take these high temperatures, and if it is a recording device, I recommend every blast furnace to get one and find out their own iron.

J. T. MACKENZIE.—What is the matter with the optical? On looking through those figures there, the optical certainly seems to be within the limits of error and it certainly is more convenient and much cheaper to use an optical of the disappearing filament type.

I was at Lynchburg a few weeks ago making tests like this on different grades of gray iron with platinum-rhodium and porcelain thermocouple, and found that whereas the correction as previously given by the makers is fully twice too much, yet once we correct for that, the optical is quite reliable for works control.

D. J. DEMOREST.—Where did you take the temperature?

J. T. MACKENZIE.—In a ladle of cast iron. The radiation type will read 900° in an absolutely clean stream, and over about 10 min. later when it has oxide on it will read about 1150°.

R. D. CARVER.—The only trouble with the optical pyrometer is that you need a man there continuously, and that means the element of personal and human error; but with the thermocouple the record is more uniform. We have had one in the roof of the open-hearth furnace for at least 4 weeks and have taken it out and examined it and there was a little difficulty there but not enough to hurt the elements. We put it back in again and it has gone at least 8 months more, showing that it is entirely practical.

We are now working on the idea of possible semicontrol at high temperatures with these thermocouples. It has worked out in this fused quartz work, as you know, the transformation point from the tridymite to the chrysophilite stage, and they want to make a material that is just on the border line, which demands very accurate control, and the only way they have been able to do it has been with this thermocouple. A difference of 5° scraps the material, and before they had this thermocouple they scrapped on the average of one out of every five or 10. Now they have been able to get every one of them perfectly. We have worked on it for some 6 or 8 years.

E. J. LOWRY.—Referring to Mr. Coleman's paper on the copper-bearing pig iron, it might be of interest to Mr. Coleman to know that one large malleable producer in

this country, investigating the effect of copper on malleabilizing, went as high as 1 per cent., but could find no difference in the malleabilizing cycle, nor in the casting practice.

W. B. COLEMAN.—That is very interesting. I have heard of a malleable foundry working up to possibly 0.5 per cent. without experiencing any trouble. It is very interesting to know you can go as high as 1 per cent.

W. E. GRIFFITHS.—Mr. Sweetser, in connection with the statement you made a while ago as to the elimination of variables and then the talk of the economic feature from the standpoint that the producer is not interested in spending a lot of time and money in investigating the virgin ore, etc., it seems to me that is the only way you can find out the influence of the various additions you make and the various elements you add, outside of carbon and iron. Start at the bottom and come up. That is the only way you can go at research work and you have to pay the bill for it.

E. J. LOWRY.—The thought was not that the producer is not interested in finding out that, but after he finds it out, what is he going to do with all this other material?

W. E. GRIFFITHS.—You might hope to find out from the investigation of the various effects of elements, such as, for instance, the effect of silicon on iron-carbon or the effect of manganese on iron-carbon, the effect that you get from those two elements. From the investigation of these simple things you might find out that you would be able to make use of these off-irons in melts of other kinds. You must get down to fundamentals first before you can do anything at all, in my opinion.

C. H. HERTY, JR.—After you get your fundamentals, you will know what to do with these things, instead of wondering beforehand what you will do with them.

W. B. COLEMAN.—Going back to the matter of copper in the pig iron, I have heard indirectly that there was some iron pipe used at one of the oil refineries that contained a certain amount of copper and resisted corrosion.

M. GRINDAL.—We have in Norway several conditions in general differing from those prevailing here. One of them is that we have our water supply under a very heavy head; particularly so on the west coast. The consequence is that we have had trouble with cast-iron pipe breaking, and we have had to look into that to some extent. Also we find trouble with corrosion. We have had water attacking the material.

I heard it stated, just by chance, that copper would reduce the tendency to corrosion. However, I was under the impression that copper also was detrimental to the physical qualities of the metal. I was very much pleased this morning to learn that this was not so, that it was not based on facts. I had the opportunity only to make a very small test in a very crude way. I took electrically-melted gray iron and poured some metal copper into the ladle, and poured some small test-pieces; I just poured some water over them and compared them with the same iron without the addition of copper. There was an enormous difference in the corrosion. It was entirely evident that copper in that case was a most effective anticorrosive. However, I am sorry to say that I did not look further into the effect, using 1 per cent. of copper, on the physical properties, but I simply wanted to mention this finding in case any one might like to go on with this investigation in this country.

You are going into research work to get at the fundamentals and bring out the facts. You are going to learn how to get the best types of pig iron. As Dr. Moldenke pointed out, we are getting back to competition and deterioration in quality. We want a cheap metal and a good metal at the same time.

I have run against the same trouble in Norway that you find here. The man who is going to have the trouble will not pay for research work so as to correct it. Why? Simply because he does not know enough about it. If you want the metal to stand the amount of service that it should, you must be willing to pay for it.

You want to buy cheaply and you want to sell cheaply, but you want to pay the proper price for the proper quality, and I think it comes down to the fact of enlightening the consumer, and that is where we lose. As engineers, as metallurgists, we are too scientific and we forget to be merchants. That is what we have to learn a little more about, and that is one of the things that I have come here to learn.

J. T. MACKENZIE.—I am glad to hear of this experience on corrosion resistance of copper. We have furnished thousands of tons of 1 per cent. copper for cast-iron pipes for use in the oil refineries and it is in the rundown lines where the condensate is first formed and the corrosion is largely hydrochloric acid, but I would like to remark that a high concentration of alkaline chlorides, even in the presence of hydrochloric acid, will to a large extent neutralize the beneficial effect of copper, and while the corrosion is enormously inhibited by copper up to 1 per cent. in pure hydrochloric or sulfuric acid, you must be very careful about the other constituents of the solution before you unhesitatingly recommend the use of copper.

L. F. REINARTZ.—I want to state that in recommending high-copper steel pipe you will experience considerable difficulty in connection with corrosion. This report comes to us from oil-well companies that have used copper pipe.

Alloy Steels

BY BRADLEY STOUGHTON,* BETHLEHEM, PA.

(Henry Marion Howe Memorial Lecture)

FROM 1886 to 1889, approximately, Henry M. Howe was engaged most actively in preparing his *Metallurgy of Steel*, which became the foundation for his leadership in the science of physical metallurgy of steel. During the same period, Hadfield was developing the first of the industrially-important alloy steels, and Riley was initiating the commercialization of nickel steel. Both of these alloys are treated briefly in Howe's book; indeed, he missed practically nothing of interest in all the steel literature preceding 1889. There had been many good books on steel before his, but none of the same character. His genius, like that of Aristotle and Francis Bacon, manifested itself in the painstaking collection of data, followed by their coordination and interpretation. Not only manganese and nickel steels were discussed, but several which most of us, with our tendency to overlook the past, are wont to regard as exclusively recent developments, such as: silicon, chrome, copper, tungsten, vanadium, titanium, chrome-tungsten and many other steels. But Howe never devoted much special attention, in his writings, to alloy steels; the science of the relations of iron and carbon offered field enough even for his capacity for constructive thought. And, today we find the science of alloy steels in much the same condition as was the metallurgy of steel before Howe wrote his first great book.

Many, many important facts have been presented to the world, especially by Sir Robert Hadfield, James Riley, Henri le Chatelier, Kotaro Honda and his associates, Maunsel White, Harry Brearley, and a host of others. Several books have been devoted wholly or chiefly to alloy steels, such as those by Hibbard, Mars, Heyn and Wetzell, Grube, Burnham, Gillett and Mack, Billón, Brearley, Monypenny, Becker, Savoia, Sauveur, Guillet, Tammann, Guertler, Goerens, Guia, Wuest, Ledebur, Schenck, Aitchison, Osann, Guettier, Buchetti, Escard. The vast number of data on the subject are indicated by a bibliography¹ which was collected in preparing this address. It covers the years 1911 to

* Consulting Metallurgist and Head of Department of Metallurgy, Lehigh University.

¹ This bibliography is to be filed, by arrangement, with the Engineering Societies' Library in New York, N. Y., where it is available to everyone, either by visit or correspondence. The author offers the opinion that bibliographies of this extent are not suitable for distribution in scientific periodicals or *Transactions*.

1926 inclusive and contains approximately 4000 references. But most of these data form merely a mass of interesting and valuable, but unco-ordinated, facts, with very little information on the fundamental principles on which these facts doubtless depend. Three hundred years ago Francis Bacon said:

No mass of material can by itself make science. Induction does not mean simple enumeration of all the data; conceivably, this might be endless, and useless. We must narrow and enclose our field in order to capture our prey. The method of induction must include a technique for the classification of data and the elimination of hypotheses; so that by the progressive cancelling of possible explanations one only shall at last remain.

Alloy steels are the basis of automobile design; they have revolutionized machine design, including locomotives; have revolutionized the art of cutting metals and other objects; the making of cutlery, the design of electro-magnetic apparatus, etc. In view of the importance of the subject and its voluminous literature, it is astonishing that no one has made a serious attempt to classify the data into one comprehensive whole from which some deductions may be drawn on the fundamental basis of the influence of the alloying elements other than carbon. It is not difficult to pick out of the mass of data the few fundamental facts which have been established or indicated; these divide themselves into two general types, of which the first seems to be connected most nearly with the physical and mechanical properties of the materials, while the second type is more fundamental still and underlies the first set. The first generalization may be illustrated by a few examples: *when a metal forms a solid solution with other metals, it gives to the alloy some of its own corrosion-resisting properties*: Thus, chromium produces "stainless iron" and "stainless steel;" nickel and copper decrease the rusting of steel; silicon gives iron some power to resist attack by acids which silicon itself resists.

Nickel.—Nickel dissolves in steel at atmospheric temperatures, and especially in the ferrite crystals in steel. The crystals are reduced in size, with resulting improvement in the combination of strength with ductility, and in the resistance to abrasion hardness.

Manganese.—Manganese, on the other hand, seems to be attracted by the carbon in steel. It forms a double carbide, $(\text{Fe, Mn})_3\text{C}$, which increases the size and hardness of the carbide crystals and thus increases hardness, brittleness and reduces strength. With 7 per cent. and more of manganese, a solid solution of iron, manganese and carbon exists at atmospheric temperature, which gives manganese steel its characteristic properties.

These generalities do not, of course, exhaust our knowledge; there are a great many other known facts, such as the effect of different alloying elements on strength, ductility, elastic limit, toughness, hardness, effect

of heat treatment, electric and magnetic properties, cutting power at different temperatures for example. These facts are contained in the 4000 references already mentioned; to gather them together would require at least one whole book. When collected, they would merely comprise what Sir Francis Bacon calls "a mass of material and not a science." A science might be built from them in time, but a more important step in this direction relates to the second type of fundamental principle mentioned above—that is to say, the constitution of the alloy steels as illustrated in their equilibrium diagrams.

THE EQUILIBRIUM DIAGRAMS OF ALLOY STEELS

Theoretically pure alloy steels, consisting of carbon, iron, and a third element, are known as ternary steels; when they have carbon, iron and two other alloying elements, they are called quaternary steels. These include chrome-tungsten, chrome-nickel, chrome-vanadium steels, and others. Vanadium must be classed as an alloying element because of its effect on constitution. Quinary steels have three alloying elements besides carbon and iron. High-speed steels are often in this category, because they contain influential amounts of tungsten, chromium and vanadium. Equilibrium models of quaternary and quinary steels are too complicated even for discussion at this time. The equilibrium models of ternary steels are, of course, the simplest, and even they require a model in three dimensions, built up from three binary equilibrium diagrams, as follows: carbon and iron; carbon and the third element; iron and the third element.

BINARY CARBON DIAGRAMS

Carbon and Iron.—The carbon-iron diagram (Fig. 1 and 2) has been very extensively studied; only that part which, at present, relates to alloy steels, namely, the iron side with less than 2 per cent. of carbon, is established without serious controversy. Recent alterations here include the delta region near the melting point of pure iron and the discontinuance of beta iron as a separate phase.

The region between 6 and 12 per cent. of carbon is still uncertain, as indicated by the discrepancy between lines in Figs. 1 and 2, both of which date from 1920. Undoubtedly, Fe_3C decomposes above a temperature of about 1700°C . This then necessitates a horizontal line at this temperature, but this has been missed by observers. Also an indentation in the liquidus must occur here, instead of at 1825° , as indicated by Ruff. There is a serious discrepancy between the solidus lines of Ruff and Ruer. to the right of the 7 per cent. carbon line.

Alloys with more than 5 per cent. of carbon may well have industrial importance, but they have not yet been made industrially because of the

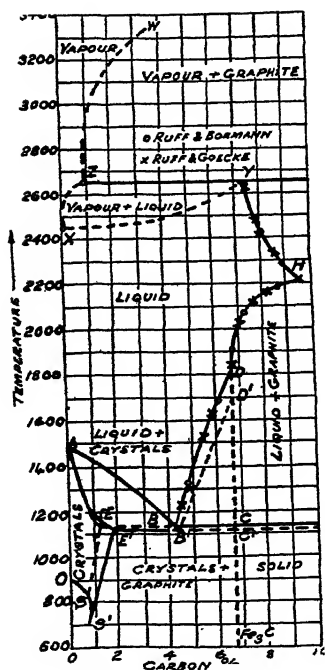


FIG. 1.—FeC (36 MM. HG) (AFTER OTTO RUFF²).

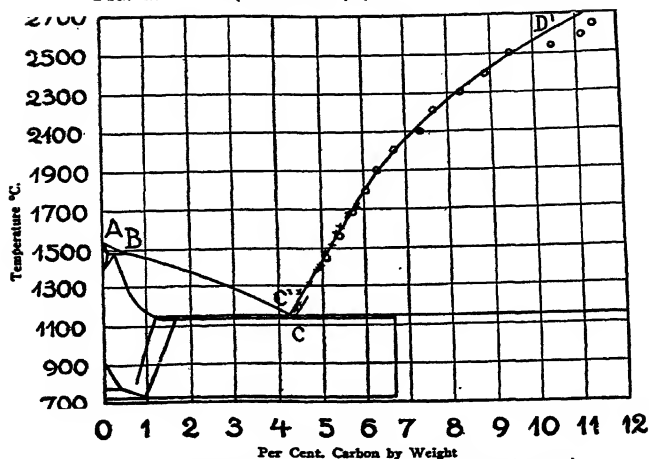


FIG. 2.—CARBON-IRON DIAGRAM (AFTER R. RUER³).

² O. Ruff: Alloys of Carbon with Manganese, Nickel, Iron, Cobalt and Chromium. *Foundry Trade Jnl.* (1920) 22, Figs. 1, 2 and 3, p. 853; Figs. 5 and 6, p. 854; from *Forschungsarbeiten*, etc.

³ R. Ruer: See G. Tammann, *Textbook of Metallography*. Chem. Cat. Co., Inc., New York (1925) Fig. 147. n. 236.

volatility of iron, which occurs at 2450° C. with pure iron⁴ and at 2650°, with alloys containing carbon.

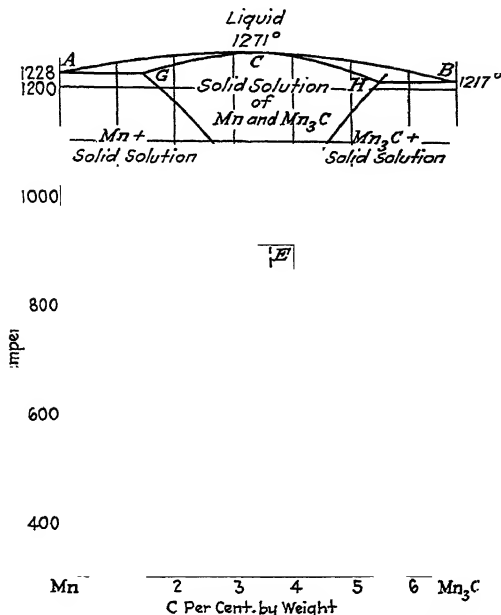


FIG. 3.—CARBON-MANGANESE DIAGRAM (AFTER K. KIDO⁶).

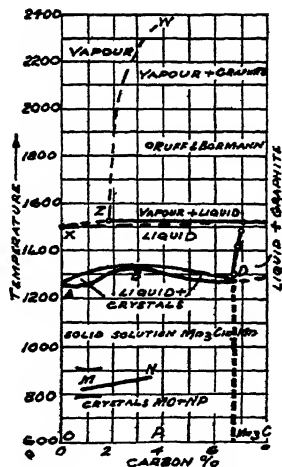


FIG. 4.—CARBON-MANGANESE DIAGRAM (AFTER OTTO RUFF⁸).

Carbon and Manganese.—Pure manganese volatilizes at 1900° C., according to *Critical Tables*,⁷ and according to Ruff (Fig. 4).⁸ This again prevents formation of alloys with more than 7 per cent. of carbon. The diagram Mn-Mn₃C, according to Kido, is shown in Fig. 3. The liquidus has a maximum of 3.5 per cent. of carbon, indicating a compound Mn₂C, but this compound does not seem to occur in the solid. At atmospheric temperature the microconstituents are seen to be very similar to

⁴ H. Landolt and R. Börnstein: *Physikalisch Chemische Tabellen*. Springer & Co., Berlin. (1923.)

⁶ K. Kido: *On the Equilibrium Diagram of Carbon-manganese Alloys*. Tohoku Univ. Sci. Rpt. (1920) 9, Fig. 3, 208.

⁸ O. Ruff: *Op. cit.*, 854.

⁷ *Elementary Substances and Atmospheric Air*. International Critical Tables. National Research Council. (1926) 102.

those of the iron-carbon alloys; in the case of manganese, however, the eutectoid occurs at 3.6 per cent. of carbon, instead of at 0.85 per cent.

Carbon and Nickel.—Again, in Fig. 5, is seen the limitation of the alloy series by the volatility of nickel.

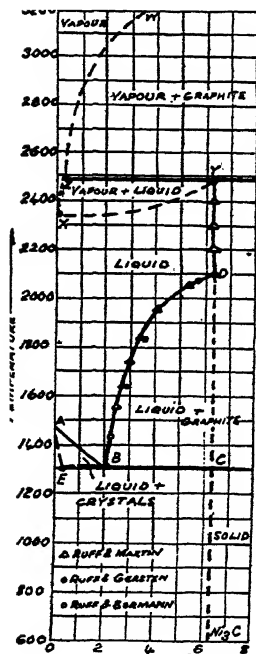


FIG. 5.—NICKEL-CARBON-DIAGRAM (AFTER OTTO RUFF⁹).

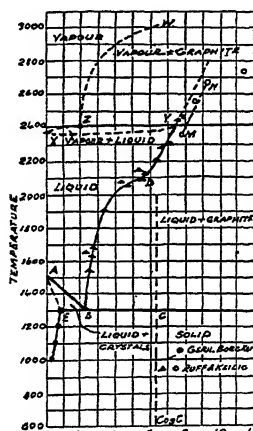
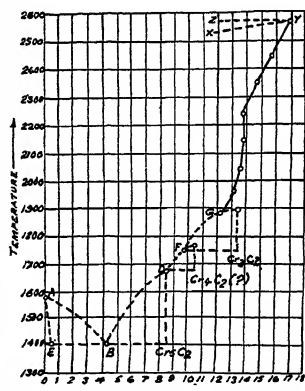


FIG. 6.—CARBON-COBALT DIAGRAM (30 MM. HG.) (AFTER OTTO RUFF¹⁰).

FIG. 7.—CARBON-CHROMIUM DIAGRAM (30 MM. HG.) (AFTER OTTO RUFF¹¹).



Carbon and Cobalt.—It is interesting to note that manganese, as well as the metals of the iron group of the periodic system of elements for which we have data, namely, nickel and cobalt (Fig. 6), all form alloys with carbon in much the same manner as does iron. The determination of the liquidus near the carbide end is very difficult for all these alloys, because of the very high temperatures involved. It is observed that the curve for cobalt is much more like the theoretical form for a decomposing compound than are those for iron, manganese and nickel. Doubtless, a horizontal line will ultimately be found below, or near, 2100° C., and an indentation where this horizontal line joins the liquidus.

^{9,10,11} O. Ruff: *Op. cit.*, 854.

Carbon and Chromium.—A series of carbon-chromium compounds occurs, instead of only one. The eutectic and the solid solution at high temperatures at the iron end are still found. It is to be noted that Ruff evidently has not sought for changes in the solid, but it would be reasonable to expect a decomposition of the solid solutions of carbon with nickel, cobalt and chromium at lower temperatures with, however, this distinction, that the solid solubility of carbon in iron and in manganese at white heat is greater than it is in nickel, cobalt and chromium at corresponding temperatures. (Fig. 7.)

Carbon and Silicon.—It is known that carbon and silicon are mutually soluble when liquid, in some concentrations. At least one compound, SiC , (commercially known as carborundum) also exists. But I have not found any determination of the equilibrium diagram of the series, nor does Honda¹² include any in his discussion of the alloys of iron-carbon-silicon referred to later.

Carbon and Copper.—Ishiwara, Yonekura and Ishigaki, in a study of the iron-carbon-copper series¹³ to which we shall later refer, assume a binary diagram of the alloys of copper with Fe_3C , but no equilibrium has been determined by experiment so far as I know.

Manganese and Iron.—Manganese and iron dissolve in each other in all proportions, both liquid and solid. There is a change of phase in the solid, but it is from the non-magnetic alloys at high temperatures to the magnetic phase below. Complete solid solubility exists in all alloys to atmospheric temperatures. (Fig. 8.) The magnetic change is indicated for alloys of 100 to 50 per cent. iron, with a short indication near 1150°C . and 100 per cent. manganese. It is to be noted that the iron-manganese solid solutions are magnetic at atmospheric temperatures; only the presence of carbon renders manganese steels non-magnetic at lower temperatures.

¹² K. Honda and T. Murakami: Structural Constitution of Iron-carbon-silicon Alloys. *Jnl. Iron Steel Inst.* (Pt. 1, 1923) 107, 545.

¹³ T. Ishiwara, T. Yonekura and T. Ishigaki: On the Ternary Diagram of the System Iron, Carbon and Copper. *Tohoku Univ. Sci. Rpt.* 15 (Apr., 1926) 81.

¹⁴ G. Rümelin and K. Fick: Beiträge zur Kenntnis des Systems Eisen-Mangan. *Ferrum* (1915) 12, Fig. 50, 43.

¹⁵ R. Ruer and E. Schütz: Das System Nickel-Eisen. *Metallurgie* (1909) 6, 679; (1910) 7, 415.

¹⁶ L. Grenet: Notes on the Iron-nickel and Iron-cobalt Equilibrium Diagrams. *Jnl. Iron Steel Inst.* (1925) 112, 272. See Fig. 6, Equilibrium Diagram of Ferro-nickels.

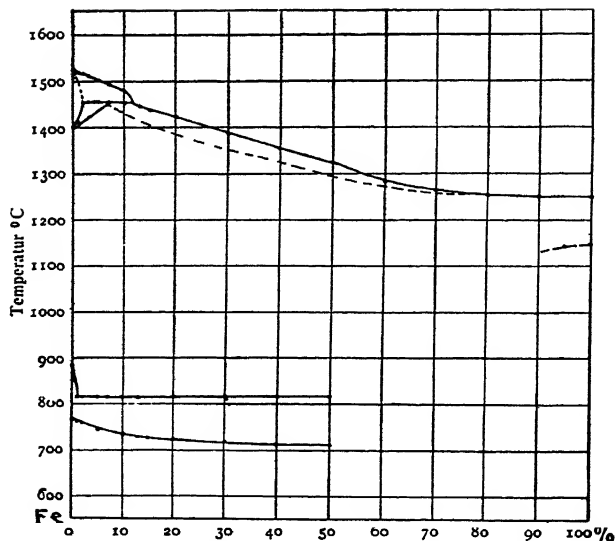


FIG. 8.—MANGANESE-IRON DIAGRAM (AFTER G. RÜMELIN AND K. FICK¹⁴).

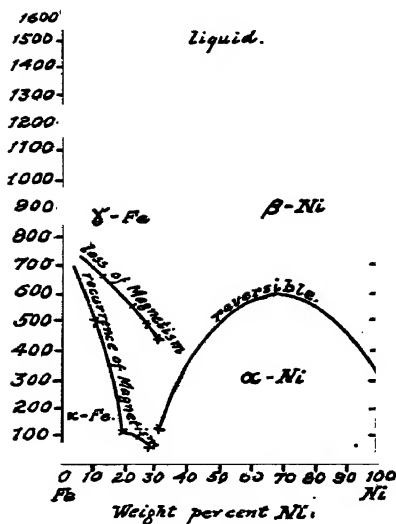


FIG. 9.—NICKEL-IRON DIAGRAM (AFTER R. RUER AND E. SCHÜTZ¹⁵).

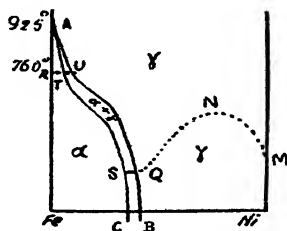


FIG. 10.—EQUILIBRIUM DIAGRAM OF IRON AND NICKEL. THE LINE SQNM IS THE CURVE OF THE MAGNETIC CHANGE OF THE GAMMA PHASE.¹⁶

Nickel and Iron.—Nickel, like manganese is soluble in iron in all proportions both liquid and solid. (Fig. 9.) The solid solutions persist to atmospheric temperatures chemically unchanged. The magnetic change in iron is affected by nickel in a manner wholly different from that which manganese produces. (Fig. 10.)

Cobalt and Iron.—Again solutions in all proportions occur for cobalt-iron (Fig. 11), both liquid and solid, with the gamma-alpha change of a new character. A later determination of this change in different alloys of the series is shown in Fig. 12.

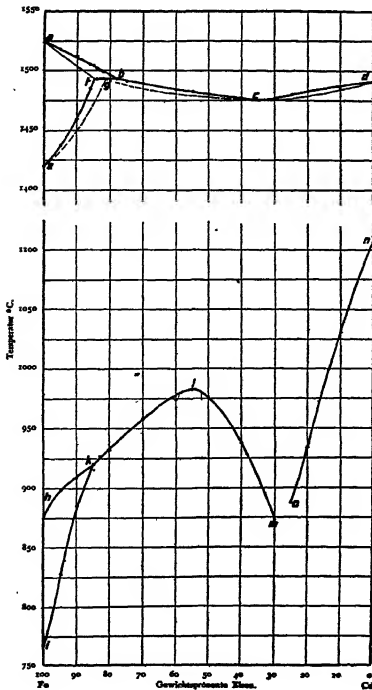


FIG. 11.—COBALT-IRON DIAGRAM AFTER R. RUER, AND K. KANEKO¹⁷).

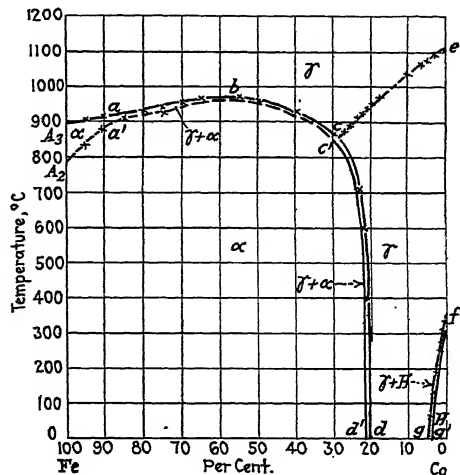


FIG. 12.—STRUCTURE DIAGRAM OF COBALT-IRON ALLOYS SHOWING GAMMA-ALPHA CHANGE (AFTER H. MASUMOTO¹⁸).

¹⁷ R. Ruer and K. Kaneko: Das System Eisen-Kobalt. *Ferrum* (1913-14) 11, Figs. 20, 35.

¹⁸ H. Masumoto: New Transformation of Cobalt and the Equilibrium Diagrams of Nickel-cobalt and Iron-cobalt. *Tohoku Univ. Sci. Rpt.*, (1926) 18, Fig. 21, 475.

Chromium and Iron.—We have only meagre details of the chromium-iron diagram, showing the formation of two solid solutions and a eutectic from the liquid alloys, but not indicating what, if any, changes occur below 1320° C. (Fig. 13.) The work of Russell¹⁹ covers chromium steels up to 12 per cent. chromium and 1 per cent. carbon, in which the carbide is shown to separate from solid solution in the general neighborhood of 700° C.

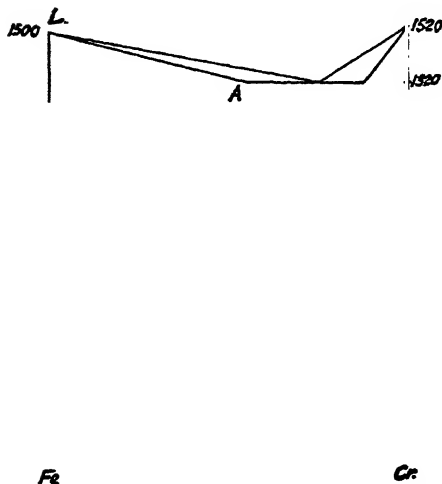


FIG. 13.—CHROMIUM-IRON DIAGRAM (AFTER T. F. RUSSELL²⁰).

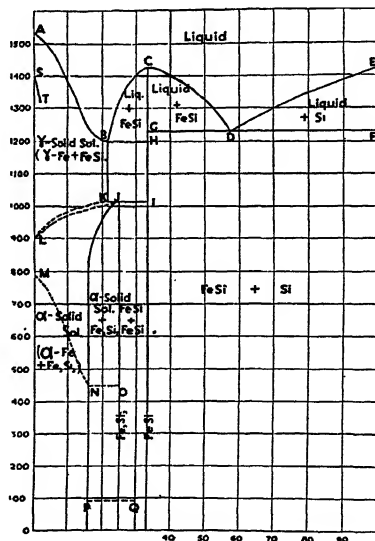


FIG. 14.—SILICON-IRON DIAGRAM (AFTER K. HONDA AND T. MURAKAMI²¹).

Silicon and Iron.—The silicon-iron diagram is one of the most interesting and complete: A stable compound, FeSi, exists and forms a liquidus-solidus and two eutectics with iron on one end and with silicon on the other (Fig. 14). A less stable compound, Fe₃Si₂, also exists at lower temperatures, but decomposes at 900° to 1020° C. The Fe₃Si₂ forms a solid solution with iron in the lower range, which becomes a solution of iron and FeSi above 900° to 1020° C. Above 33 per cent. Si, the alloys form a regular eutectic series of FeSi with silicon.

^{19,20} T. F. Russell: On the Constitution of Chromium Steels. *Jnl. Iron Steel Inst.* (1921) 104, 247.

²¹ K. Honda and T. Murakami: *Jnl. Iron Steel Inst.* (1923) 107, Fig. 1, 547.

Copper and Iron.—Copper dissolves in iron to a maximum of 26 per cent. in liquid; iron dissolves in copper in the liquid state to a maximum of 29 per cent. The liquid solubility on the copper end decreases rapidly with falling temperature below 1440°C. , until, at 1094° , copper holds only 2 to 3 per cent. of iron in liquid solution. The solid solution on this end contains about 3 per cent. iron, which decreases with further fall in temperature, as indicated by line *mqu*. On the iron end, the solid solution begins with about 21 per cent. copper at 1440° , decreases, as shown by the dotted line *kpt*, (Fig. 15) until it is scarcely 1 per cent. at atmospheric temperatures. The solid alloys at atmospheric tempera-

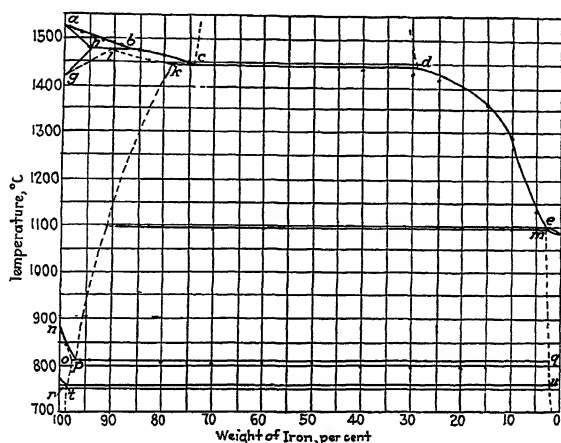


FIG. 15.—COPPER-IRON DIAGRAM (AFTER R. RUER AND K. FICK²²).

ture consist of crystals of iron with a few traces of copper in solid solution and crystals of copper containing less than 3 per cent. iron in solid solution.

Tungsten and Iron.—The tungsten-iron series includes one chemical compound, Fe_3W_2 , which, like Fe_3Si_2 and Fe_3C , decomposes on heating. This decomposition is indicated by the horizontal line at 1640°C. (Fig. 16). A very limited solid solution of this Fe_3W_2 in tungsten exists at the tungsten end of the series, as indicated, and a much more extensive solid solution at the iron end. The determination of these equilibria represents very skillful scientific work at high temperatures, as any metallur-

²² R. Ruer and K. Fick: *Das System Eisen-Kupfer. Ferrum* (1913) 11, Fig. 60, 49.

gist will recognize. The diagram is checked in many details by an earlier research, which is well worth including here for comparison (Fig. 17). The horizontal lines at 1030° and at 790° are probably both present and escaped observation by the instruments used by Sykes.

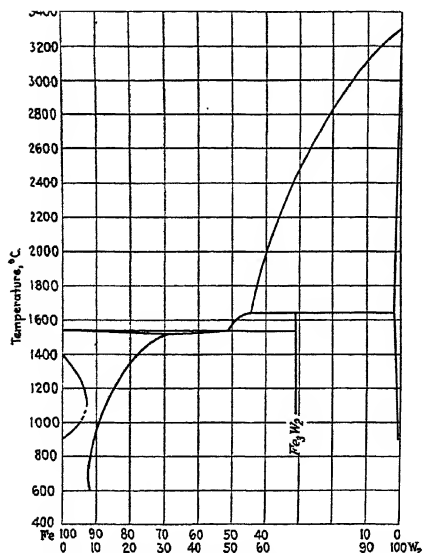


FIG. 16.—TUNGSTEN-IRON DIAGRAM (AFTER W. P. SYKES²³).

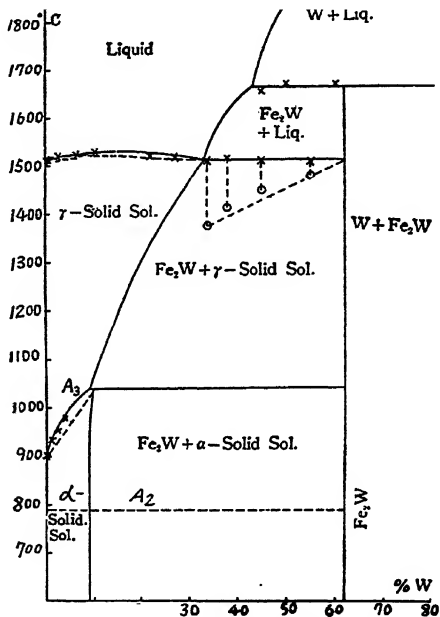


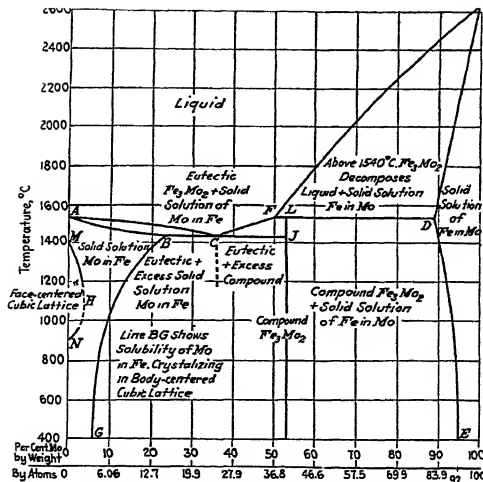
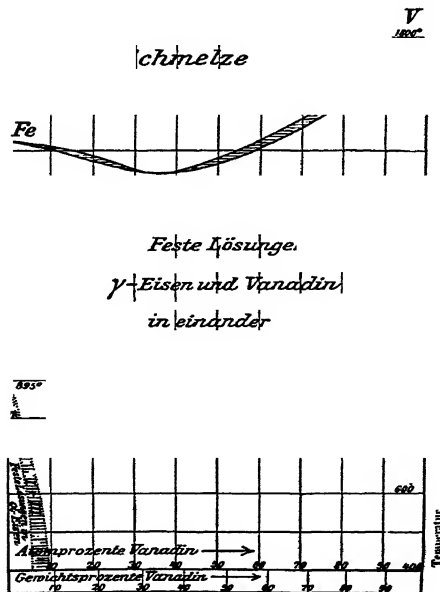
FIG. 17.—EARLIER TUNGSTEN-IRON DIAGRAM (AFTER S. OZAWA²⁴).

Molybdenum and Iron.—Sykes, by another brilliant piece of investigating work, has also given us a diagram of the molybdenum-iron alloys. (Fig. 18.) These are very similar in constitution to the tungsten-iron alloys, with the principal distinction that they include a more extensive solid solution of iron in molybdenum, and that the pasty stage near the iron end is more pronounced.

Vanadium and Iron.—These metals form complete mutual solutions both liquid and solid. There is a compound indicated at 31.5 per cent. vanadium, corresponding to the formula, Fe_3V . This does not appear in the alloys, however, but complete solid solution extends for all concentrations from the solidus to atmospheric temperature. Vanadium depresses the magnetic change in iron, and (by extrapolation) alloys with more than 12 per cent. vanadium are non-magnetic at atmospheric temperatures. (Fig. 19.)

²³ W. P. Sykes: The Iron-tungsten System. *Trans.* (1926) **73**, Fig. 3, 971.

²⁴ S. Ozawa: On the Constitutional Diagram of the Iron-carbon-tungsten System. *Tohoku Univ. Sci. Rpt.* (1922) **11**, Fig. 4, 339.

FIG. 18.—MOLYBDENUM-IRON DIAGRAM (AFTER W. P. SYKES²⁵).FIG. 19.—VANADIUM-IRON DIAGRAM (AFTER W. GUERTLER²⁶).

²⁵ W. P. Sykes: The Iron-molybdenum System. *Trans. Am. Soc. Steel Treat.* (1926) 10, 839.

²⁶ W. Guertler: *Metallographie*. Borntraeger Bros., Berlin. (1921) 1, Fig. 171, 388.

Zinc and Iron.—The zinc-iron alloys are important because of the practice of galvanizing. The solid solution of zinc in iron, which extends to 19 per cent. of zinc (Fig. 20), forms an adhering coating which protects

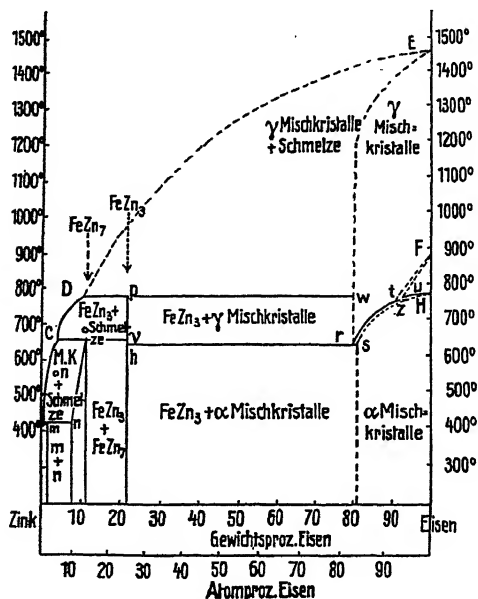


FIG. 20.—ZINC-IRON DIAGRAM (AFTER U. RAYDT AND G. TAMMANN²⁷).

iron from rusting. The features of the equilibria are clear from the diagram. There are two compounds, both of which decompose on heating.

GENERAL CONCLUSIONS ON THE BEHAVIOR OF BINARY ALLOYS

About 20 other binary iron diagrams are on record, and it occurred to me that some interesting generalities might be indicated if the main features of these were arranged in one table, classified according to the periodic system of chemical elements. The data are still too meager to be conclusive, but some generalities are at least indicated as far as the known facts go. (In speaking of Group VII it is to be noted that we have only one example from which to generalize (?) in that group.) The indications on the table as a whole are:

Liquid Solubility.—Liquid iron is mutually soluble in all proportions with most of the liquid elements of its own group and of Groups II, III,

²⁷ U. Raydt and G. Tammann: *Struktur und Eigenschaften von unter Druck zusammengeschmolzenen Zink-Eisenlegierungen. Ztschr. anorg. Chem.* (1913) **83**, Fig. 4, 265.

VI and VII. The tendency is towards liquid insolubility with elements of Group I, and with the heavier atoms in Groups IV (the position of cerium in this group is debatable) and V.

Hot Solid Solubility.—Solid iron is mutually soluble in all proportions at high temperatures with solid metals of its own group and Group VII. The tendency is for partial mutual solubility at high temperatures with elements in Groups I, II and VI. The tendency is for iron to dissolve

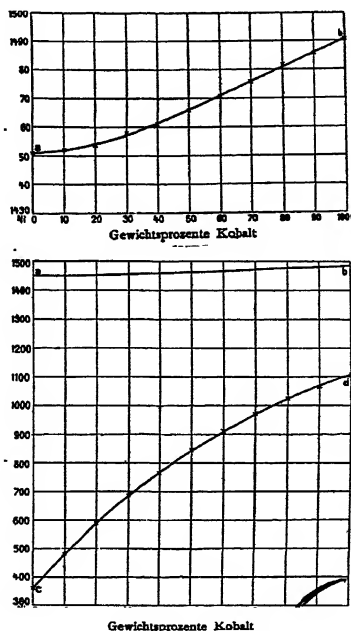


FIG. 21.—COBALT-NICKEL DIAGRAM (AFTER H. MASUMOTO^{28,29}).

limited proportions of the elements of Groups III and IV in the solid state at high temperatures; but the tendency for iron to dissolve in them is not so well indicated. In elements of Group V the tendency is variable.

Cold Solid Solubility.—At atmospheric temperatures iron is mutually soluble in all proportions with metals of its own group and Group VII. With the elements in Groups I, II and VI there seems to be lessened mutual solid solubility at atmospheric than at high temperatures, and a still greater decrease with elements in Groups III (?), IV and V.

²⁸ H. Masumoto: On a New Transformation of Cobalt and the Equilibrium Diagrams of Nickel-cobalt and Iron-cobalt. *Tohoku Univ. Sci. Rpt.* (Oct., 1926) 15, 449-477.

²⁹ R. Ruer and K. Kaneko: Das System Nickel-Kobalt. *Metallurgie.* (1912) 9, Figs. 414, 415, p. 420.

TABLE 1.—*Chemical Equilibria in Iron Alloys*

Refer- ence No.	Group	Ele- ment	Atomic		Liquid Solubility	Solid Solubility—Hot	Eutectics	Compounds	Solid Solubility—Cold
			Weight	Num- ber					
1 } 2 } 3 }	(Iron)	Co	59	27	Complete	Throughout series	No		Throughout series
4 }		Ni	59	28	Complete	Throughout series	No		Throughout series
5 }		Pt	195	78	Complete	Throughout series	No		Throughout series
4 } 5 } 6 }	I	Cu	64	29	Partly immiscible	Slight; at each end	No	No	Slight; at each end
7 }		Ag	108	47	Wholly immiscible ¹	No alloy	No	No	Lesser; ² At each end
8 }		Au	197	79	Complete	$\frac{3}{4}$; one at each end	One	No	Limited
9 }	II	Be	9	4	Complete	For limited portion	One	One	At both ends
10 }		Mg	24	12	Complete	Yes	One ?	Two	?
11 }		Zn	66	30	Complete	Probably	One	One	?
12 }	III	Cr	112	43	In part only ³				
13 }		Hg	201	80	In part only ³				
14 }		B	11	5	Complete	Slight; at iron end	One	One	Eutectoid. Only known to 9 per cent. B
15 }	IV	Al	27	13	Complete	At iron end	One	One	At iron end to 30 per cent. Al
16 }		C	12	6	In part only ⁴	Minute; at iron end	One	One	Eutectoid. Only known to 7 per cent. C
17 }		Si	28	14	Complete	At iron end	Two	Two	Eutectoid.
18 }	(?)	Ti	48	23	Complete	Minute; one or both ends	One	One	?
19 }		Zr	91	40	Complete	Iron up to 18 per cent. Sn	One (?)	Several	?
20 }		Sn	119	50	Partly immiscible	Iron up to 12 per cent. Co	One	Two	Iron up to 19 per cent. Sn
21 }	V	Ce	140	58	Complete	Practically no alloy			
22 }		Pb	207	82	Immiscible ¹				
23 }		Th	232	90	Immiscible				
24 }	VI	N	14	7	Complete	Minute; at iron end	One	Two	Minute; at iron end. (Only known to 22 per cent. P).
25 }		P	31	15	Complete	Throughout series	No	One (?)	Throughout series
26 }		V	51	23	Complete	Small; at iron end	Two	Three	Yes. Known only to 57 per cent. As
27 }	VII	As	75	33	Complete	Minute; at both ends	Two	Three	Minute; at both ends
28 }		Sb	120	51	Complete	No alloy			
29 }		Bi	209	83	Wholly immiscible ¹				
30 }	VIII	S	32	16	Complete	No	One	One	Only Fe with FeS
31 }		Cr	52	24	Complete	Large; at each end	One	No(?)	At each end
32 }		Mo	96	42	Complete	At each end	One	One	At each end
33 }	IX	W	184	74	Complete	At each end	Probably	One	No
34 }		U	238	92	Complete	No			
35 }		Mn	55	25	Complete	Throughout series	No		Throughout series

¹ When the terms "immiscible" and "wholly immiscible" are used, it is to be understood that the intent is practical, not theoretical. We count two metals immiscible if they form two fluid layers, even though each layer might contain a trace of the other metal in solution.

² The word "lesser" here indicates that the solid solution in the cold is lesser in extent than the solid solution at higher temperatures.

³ The two series of alloys of iron with cadmium and mercury have never been completed, because these metals volatilize when added to liquid iron for the purpose of forming solutions.

⁴ Liquid solutions of carbon can only be made up to about 7 per cent. carbon, because of the volatility of iron at 2450 to 2650° C.

⁵ Large at iron end and small at tungsten end.

Reference Numbers:

1. *Iron-cobalt*. R. Ruer and K. Kaneko: Das System Eisen-Kobalt. *Ferrum* (1913-14) 11, 33, 254.
- R. Ruer and K. Kaneko: Über eine bei konstanter Temperatur verlaufende, mit starker Wärmetönung und starker Änderung der Magnetisierbarkeit verknüpfte polymorphe Umwandlung. *Phys. Zeit.* (1914) 15, 17.
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- R. Ruer and E. Schütz: Das System Eisen-Nickel. *Metallurgie* (1910) 7, 415.
3. *Platinum*. E. Isaac and G. Tammann: Über die Legierungen des Eisens mit Platin. *Ztschr. anorg. Chem.* (1907) 55, 63.
4. *Copper*. E. Isaac and G. Tammann: Über die Legierungen des Eisens mit Zinn und Gold. *Ztschr. anorg. Chem.* (1907) 53, 281.
5. *Silver*. G. J. Petrenko: Über die Legierungen des Silbers mit den Metallen der Eisengruppe. *Ztschr. anorg. Chem.* (1907) 53, 215.
6. *Gold*. M. A. Portevin: Les alliages d'or. *Rev. Met.* (Mem.) 5, 182.
7. *Beryllium*. G. Oesterheld: Über den Schmelzpunkt und die Schmelzwärme des Berylliums. *Ztschr. anorg. Chem.* (1916) 97, 6.
8. *Zinc*. E. Müller and G. Grube: Das Eisen und Seine Verbindungen und die Legierungen des Eisens. (1917) Fig. 15, 471. Theodor Steinkopff, Dresden.
9. *Cadmium*. E. Müller and G. Grube: *Op. cit.* 474. (Eisen-Cadmium.)
10. *Mercury*.
11. *Boron*. G. Hannesen: Über Borstähle. *Ztschr. anorg. Chem.* (1914) 89, 257.
12. *Aluminum*. L. Guillet: Diagrammes des alliages binaires d'aluminium. *Rev. Met.* (1921) 18, 500.
- G. Masing and O. Dahl: Über die Ausdehnung bei der Erstarrung von eisenhaltigen Aluminium. *Ztschr. anorg. Chem.* (1926) 154, 191.
13. *Carbon*. O. Ruff: Alloys of Carbon with Manganese, Nickel, Iron, Cobalt and Chromium. *Foundry Trade Jnl.* (1920) 22, 853.
14. *Titanium*. K. Tamaru: On the Equilibrium Diagram of the System Iron-carbon-titanium. Tohoku Univ. Sci. Rpt. (1925) 14, 25.
15. *Zirconium*. Not found.
16. *Tin*. E. Isaac and G. Tammann: Über die Legierungen des Eisens mit Zinn und Gold. *Ztschr. anorg. Chem.* (1907) 53, 281.
17. *Cerium*. L. Guillet: Les alliages de Cérium. *Rev. Met.* (Mem.) (1922) 19, 352.
18. *Lead*. W. Guertler: *Metallographie*. (1912) 1, 578. (Fig. 215.) Borntraeger Bros., Berlin.
19. *Thallium*. W. Guertler: *Metallographie* (1912) 1, 575. (Fig. 214.)
20. *Nitrogen*. Not found.
21. *Phosphorus*. E. Müller and G. Grube: *Op. cit.*, Fig. 21, 486.
22. *Vanadium*. R. Vogel and G. Tammann: Über Vanadin-Eisenlegierungen. *Ztschr. anorg. Chem.* (1908) 53, 79.
- P. Fäts: Einfluss des Vanadiums auf Eisen und Stahl. *Metallurgie* (1906) 3, 635.
23. *Arsenic*. K. Friedrich: Eisen und Arsen. *Metallurgie* (1907) 4, 129.
24. *Antimony*. N. S. Kurnakow and N. S. Konstantinow: Antimonide des Eisens und des Cadmiums. *Ztschr. anorg. Chem.* (1908) 53, 1.
25. *Bismuth*. E. Isaac and G. Tammann: Über das Verhalten von Eisen zu Blei, Wismut, Thallium und Cadmium. *Ztschr. anorg. Chem.* (1907) 55, 58.
26. *Sulfur*. K. Friedrich: *Metallurgie* (1910) 7, 257.
- Iron sulfide (FeS). E. Müller and G. Grube: *Op. cit.*, 494.
27. *Chromium*. T. F. Russell: Constitution of Chromium Steels. *Jnl. Iron Steel Inst.* (1921) 104, Fig. 18, 286.
28. *Molybdenum*. E. Müller and G. Grube: *Op. cit.*, 498.
- M. A. Portevin: Les alliages pseudobinaires. *Rev. Met.* (Mem.) (1911) 8, 7.
29. *Tungsten*. W. P. Sykes: The Iron-tungsten System. *Trans.* (1926) 73, 968.
30. *Uranium*. E. P. Polushkin: Alloys of Iron and Uranium. *Carnegie Res. Mem.* (1920) 10, 129.
31. *Manganese*. G. Rümelin and K. Fick: Beiträge zur Kenntnis des Systems Eisen-Mangan. *Ferrum* (1915) 12, 41.

Almost all these diagrams will also be found collected in the double pamphlet by Müller and Grube, entitled: "Das Eisen und Seine Verbindungen," by Erich Müller and "Die Legierungen des Eisens" by G. Grube. Published by Theodor Steinkopff, Dresden and Leipzig, 1917.

Eutectic Ranges.—Eutectics occurring over a range of concentrations, do not occur in alloys with metals of the iron group or Group VII. This is obvious. They occur rather generally in all other iron alloys.

Chemical Compounds.—Compounds between iron and metals of its own group and Group VII would obviously be absorbed or hidden, if present. They tend to occur in all the alloys that are not mutually soluble in all proportions liquid and solid, except those with elements in Group I.

OTHER BINARY ALLOYS

A comparison of alloys of metals of the iron group with one another, or with elements other than iron, is interesting. For example, Fig. 21 shows how similar is the formation of mutual solid solutions in

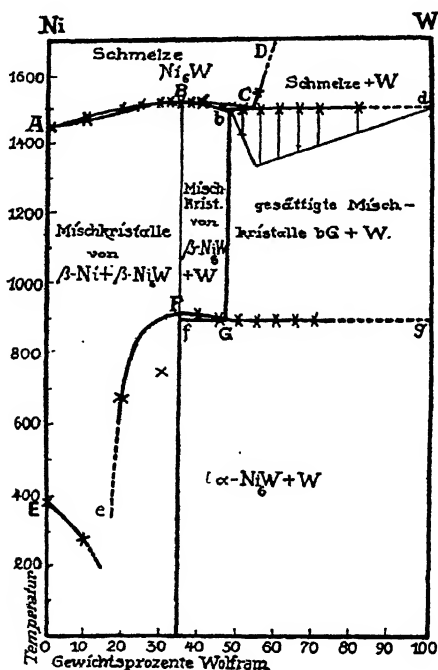


FIG. 22.—TUNGSTEN-NICKEL DIAGRAM (AFTER R. VOGEL³⁰).

alloys of nickel with cobalt to alloys of both of these metals with iron (Figs. 9 and 11). But, in respect to magnetic changes in the solid solutions, there are important differences. Parenthetically, the line running

³⁰ R. Vogel: Über Wolfram-Nickellegierungen. *Ztschr. anorg. Chem.* (1921) 116, Fig. 1, 235.

from letter *e* in Fig. 21 illustrates a new transformation in cobalt, and may be worthy of further investigation, in view of the apparent importance of cobalt in increasing the effectiveness of permanent magnets, high-speed steel and cutlery.

The alloys of nickel with tungsten (Fig. 22) bear some resemblance to the alloys of iron with tungsten (Figs. 16 and 17).

TERNARY IRON ALLOYS

An early method of illustrating the constitution of commercial alloy steels was to construct a figure similar to Fig. 23. This is a good and usual means of illustrating the micro-constituents of alloy steels at atmospheric temperature, which have been cooled as normally occurs during industrial manufacture. It does not represent equilibrium conditions. It aids materially in comprehending the reasons for several observed physical

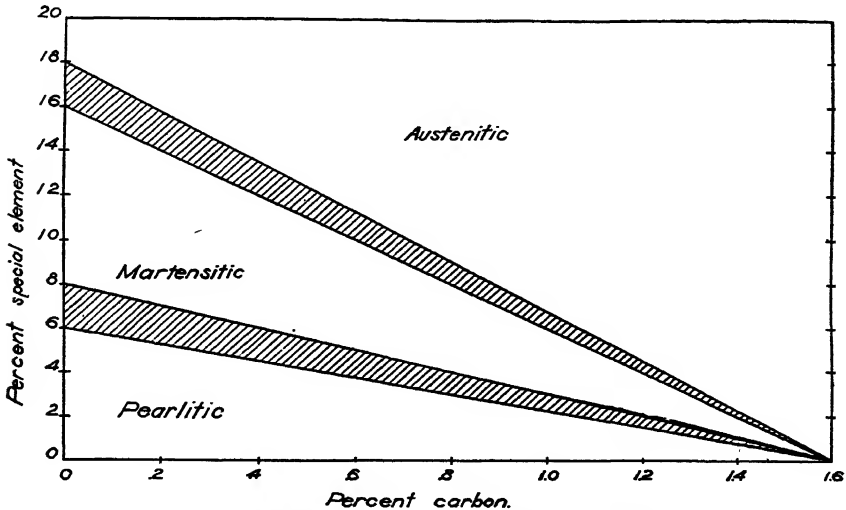


FIG. 23.—CONSTITUTIONAL DIAGRAM OF SPECIAL STEELS (AFTER A. SAUVEUR³¹).

properties of alloy steels of different proportions of three components. But to get a comprehensive view, or to grasp the possibilities, of any system of three varying components, the usual method is to erect binary diagrams along the three sides of a triangle, which sides can then severally represent the composition of the binary alloys, while the ordinates vertical thereto represent temperatures. Any point in the interior of the triangle

³¹ A. Sauveur: The Metallography and Heat Treatment of Iron and Steel. McGraw-Hill Publishing Co., Inc. (3d ed., 1926) Fig. 277, Constitutional diagram of alloy steels, 308.

would then represent the composition of the respective alloy in three components.

Such a ternary diagram is shown in Fig. 24, by Ishiwara, Yonekura and Ishigaki. As the alloys of copper with carbon have not been studied experimentally, the authors have constructed a theoretical diagram of Cu-Fe₃C. The result forms a very interesting ternary diagram, so interesting that it may justify us in disregarding the doubtful expedient of assuming that copper forms a series of alloys with Fe₃C, or of neglecting

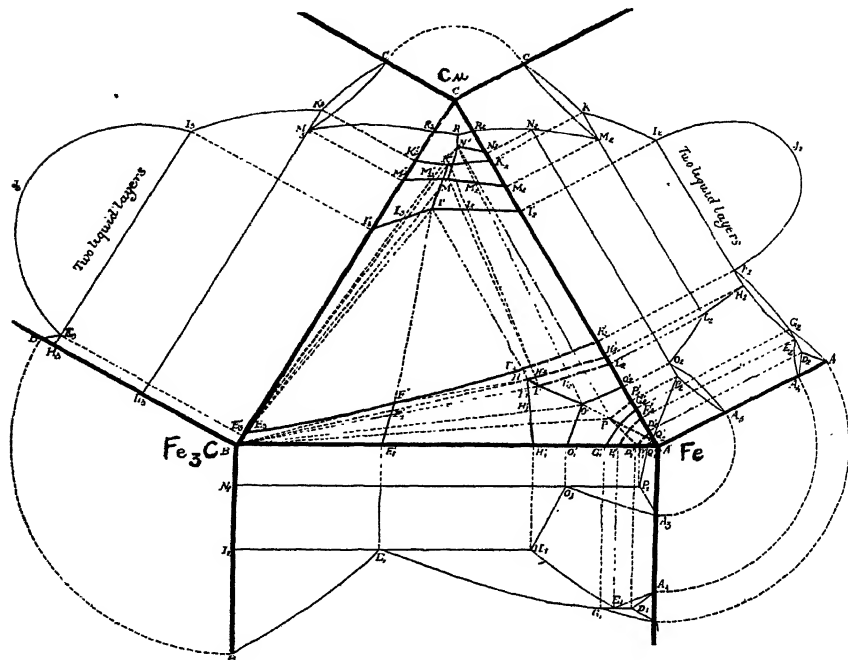


FIG. 24.—CONSTITUTIONAL DIAGRAM OF TERNARY ALLOY OF IRON (AFTER T. ISHIWARI, T. YONEKURA AND T. ISHIGAKI³²).

the very important discontinuity of the iron-carbon diagram near Fe₃C. Both of these assumptions were necessary in order to build even a tentative complete diagram.

The three binary diagrams having been set in place, the next step is obviously to make cooling curves of three-part alloys in the interior of the triangle, and plot them in their appropriate positions. This can be done by drawing lines representing composition and indicating by shading the changes in the alloy. Such a method is shown in Fig. 33. Another

³² T. Ishiwari, T. Yonekura and T. Ishigaki: *Op. cit.*, Fig. 5, p.86.

way is to swing the three binary diagrams upward on the sides of the triangle as axes and then join them in a skeleton model, as shown in Fig. 25. The interior parts may then be constructed by experiment. But the extent of the Cu-Fe-C alloys is so small, owing to the insolubility of liquid copper in liquid iron and in iron-carbide (?), that the skeleton model gives very little further information than was afforded by the binary diagrams.

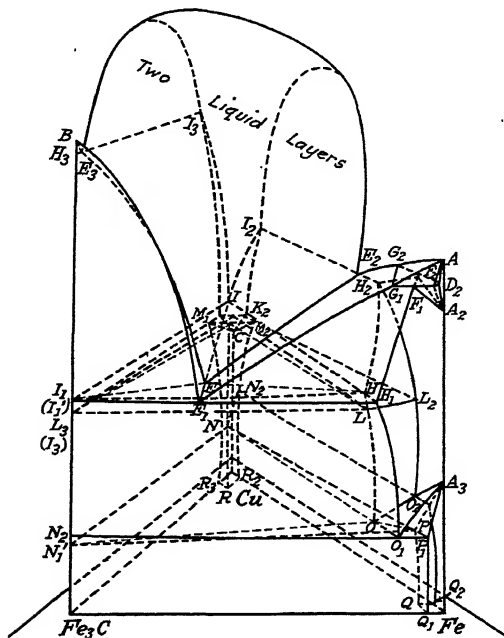


FIG. 25.—SKELETON MODEL OF THE THREE BINARY DIAGRAMS OF THE COPPER-IRON-CARBON ALLOYS (AFTER T. ISHIWARI, T. YONEKURA AND T. ISHIGAKI³³).

Another ternary diagram emanating from Honda's laboratory is shown in Fig. 26. This is a product of the same research to which we owe the facts illustrated in Fig. 14. It determined a number of interesting facts, which are set forth in Honda's conclusions.³⁴ But the handicap of a diagram like this is that it is difficult for one to grasp who is not constantly engaged in this field of investigation.

³³ T. Ishiwari, T. Yonekura and T. Ishigaki: *Op. cit.*, Fig. 4, p. 85.

³⁴ K. Honda and T. Murakami: *Op. cit.*, 581.

(solid solutions) are shown in green: the two-phase portions in the solid are shown in yellow and green, or blue and green, or yellow and blue, as the case may be, to indicate the several phases.

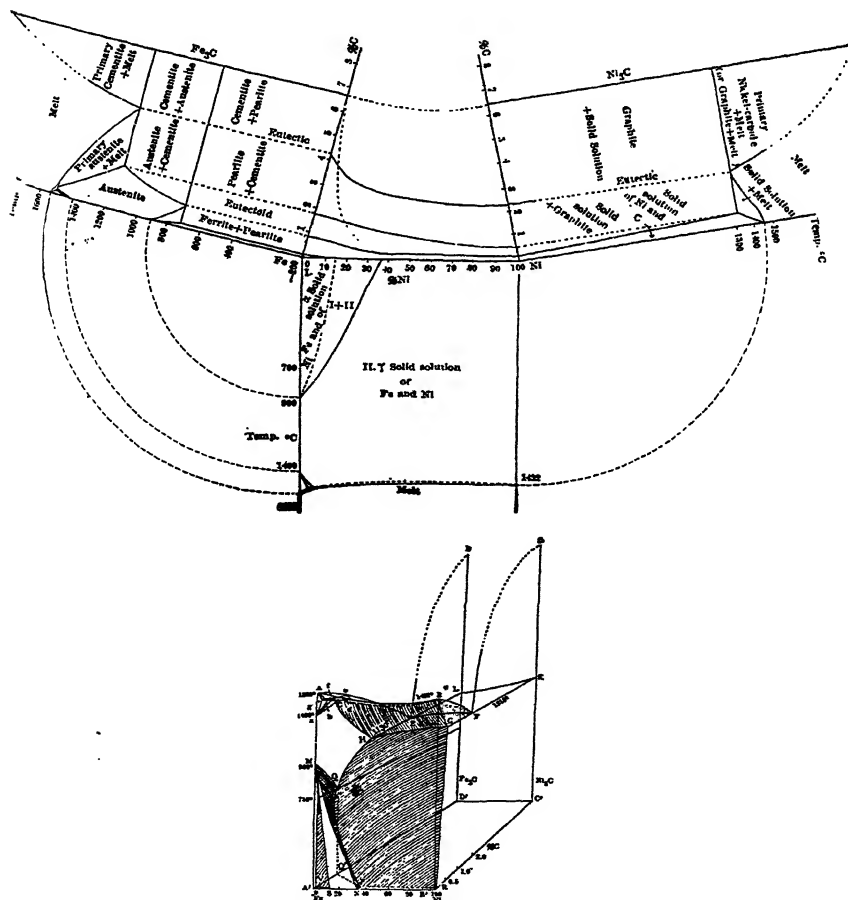


FIG. 27.—EQUILIBRIUM DIAGRAM AND SPACE MODEL OF NICKEL-CARBON-IRON ALLOYS (AFTER T. KASÉ²⁶).

A wire model has also been built to illustrate better the constitution of the alloy system. In making the model, a larger unit is used in plotting the diagrams of Fe-Fe₃C and Mn-Mn₃C, than in that for Fe-Mn, for obvious reasons. This makes the model in the form of an isosceles

²⁶ T. Kasé: Equilibrium Diagram of the Iron-carbon-nickel System. Tohoku Univ. Sci. Rpt. 14 (1925). Figs. 27 and 28, 173.

triangle, similar to that in Fig. 27. The binary diagrams were drawn from known data; I also hoped to find some data for sections containing the three components in sufficient detail to permit drawing sections for the interior of the triangle, but this hope proved groundless, and this work I hope to do myself at some future time.

The wire model was made for use at this lecture; it lends itself very well to the study and demonstration of the constitution of ternary alloys,

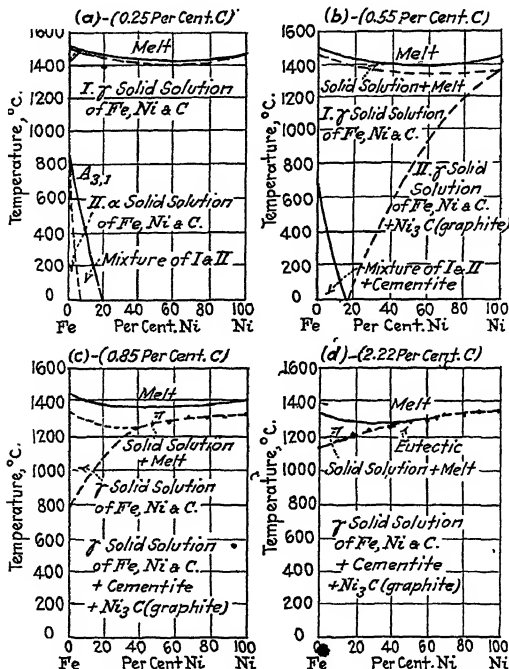


FIG. 28.—SOME SECTIONS THROUGH THE INTERIOR OF THE NICKEL-CARBON-IRON TRIANGLE (AFTER T. KASÉ³⁷).

but its reproduction in print does not show the different points clearly. If all points could be determined, and if the system could be completed in the form of an equilateral triangle, the reproduction as an illustration might be intelligible, as indicated by Figs. 30 to 34. But the primary purpose of a wire model of ternary alloys would seem to be to clarify the study of a system during a research. Wire models have been used to illustrate the constitution of ternary diagrams by Guertler and by

³⁷ T. Kasé: *Op. cit.*, Fig. 29.

Rosenhain.³⁸ For assistance in constructing the one shown herewith, I am greatly indebted to my associate, Gilbert E. Doan, who learned the method under Guertler. This model is made of zinc wire, which lends itself excellently to the purpose, because it combines pliability with adequate stiffness. The pieces are cut and bent to the shape of the line they are to represent, and are then soldered together at junction points without difficulty.

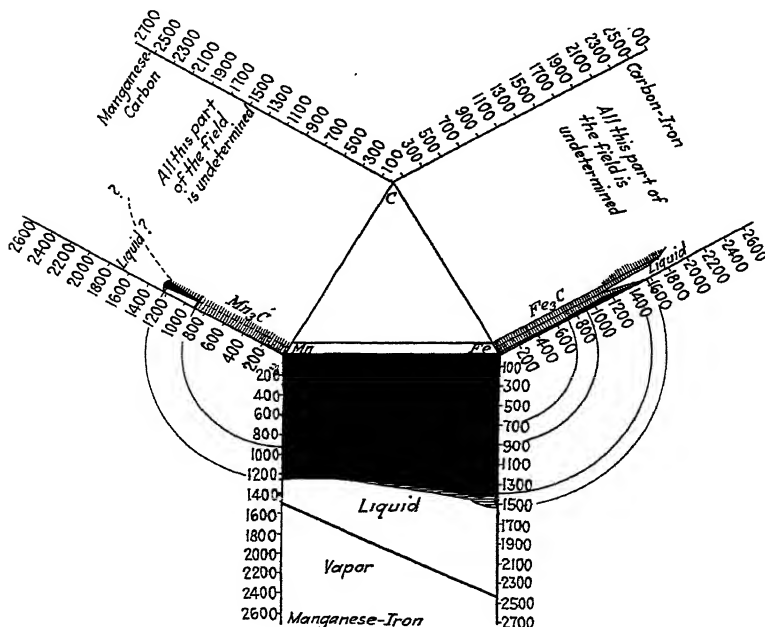


FIG. 29.—TERNARY DIAGRAM OF THE MANGANESE-CARBON-IRON ALLOYS.

Rosenhain's form of wire model is shown in Fig. 30. He says of it, "The value of the model lies in the fact that it permits the simultaneous viewing of a number of different intersecting and superposed surfaces." He also mentions the difficulty of gaining a clear picture of such an illustration by means of photographs.

For illustrative reproduction, plaster models may be made from the wire model, or shadowgraph drawings may be made. This is indicated in Fig. 31, which is the liquidus surface, and Fig. 32, which is the solidus surface. Obviously, shadowgraphs enough must be made to illustrate

³⁸ W. Rosenhain: Model for Representing the Constitution of Ternary Alloys. *Jnl. Inst. Metals* (1920) **23**, 247.

all significant surfaces, not only during the freezing of the ternary alloys, but for all important changes in the solid state. However, for study during a research, the model gives a very comprehensive picture of the change points. It would doubtless be desirable to paint the wires different colors, in order to illustrate more strikingly the different phases: A single liquid phase would be painted white; a pasty stage, white with a second color in stripes; a single solid phase, green; a solid two-phase portion, born from the single phase, would be yellow and blue; etc.

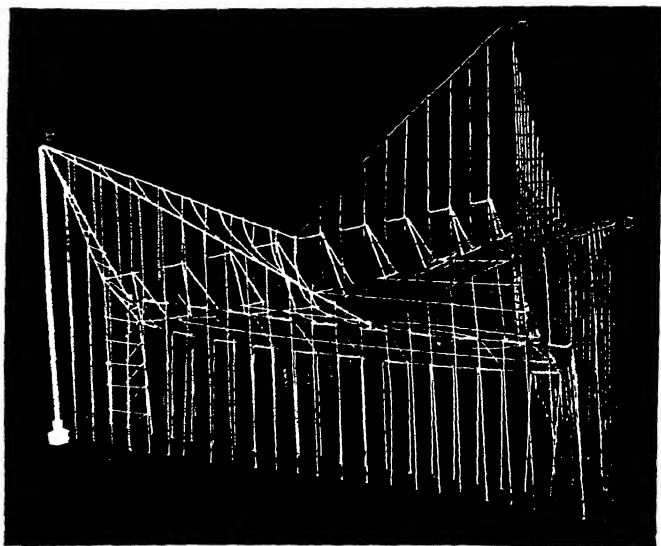


FIG. 30.—PHOTOGRAPH OF A WIRE MODEL OF THE STRUCTURE OF A TERNARY ALLOY (AFTER W. ROSENHAIN³⁹).

Fig. 33 shows three methods of illustrating the liquidus of a series of ternary solutions. It does not seem to me that the isotherm and the shading method of indicating the temperature lines of a surface, shown in Fig. 33, *a* and *b*, is as stimulating to thought as are the wire model, the plaster model and the shadowgraph.

Guertler has probably done more than anyone to develop the use of wire models for ternary alloys,⁴⁰ and to clarify the study and presentation,

³⁹ W. Rosenhain: *Op. cit.*, Fig. 1, 250.

⁴⁰ W. Guertler: *Forschungsarbeiten zur Metallkunde*. Pt. 8, *Sechs Vorlesungen zur Einfuehrung in das Verstaendnis der modernen Spezialstaele*. Borntraeger Bros., Berlin (1922).

of the subject of alloy steels. One of his methods of presentation is illustrated in Figs. 34 and 35. Figure 34 shows a wire model of a ternary steel, ideal for the reason that no complete series has ever been determined. This shows the three binary alloys, each with a eutectic, the ternary eutectic, and a solid solution forming in each corner of the triangle. No changes in the solid are indicated. On this drawing Guertler then lays a piece of transparent paper and indicates the phase which he wishes to emphasize. The transparent paper drawing may then be studied by

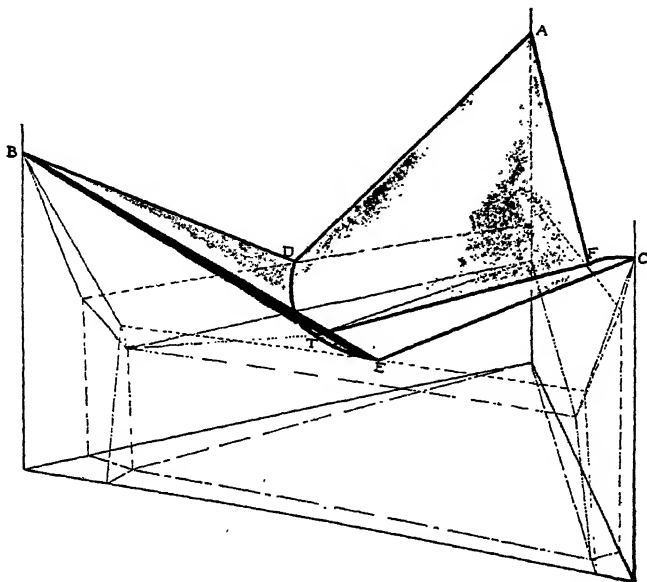


FIG. 31.—SHADOWGRAPH OF THE LIQUIDUS SURFACE OF A TERNARY ALLOY MADE FROM THE WIRE MODEL (AFTER W. ROSENHAIN⁴¹).

itself, or may be superimposed on the full model. Such a disposition is shown in Fig. 35, where the three solid solutions in the corners of the system are indicated. Other structures may be progressively studied in the same manner. In the monograph to which reference is here made, Guertler uses 18 pieces of transparent paper to indicate different parts of the model. Other methods of studying and illustrating ternary alloys have been proposed; they are included in the bibliography already mentioned. None of them seem to me to be as promising as those just described.

⁴¹ W. Rosenhain: *Op. cit.*, Fig. 2, 251.

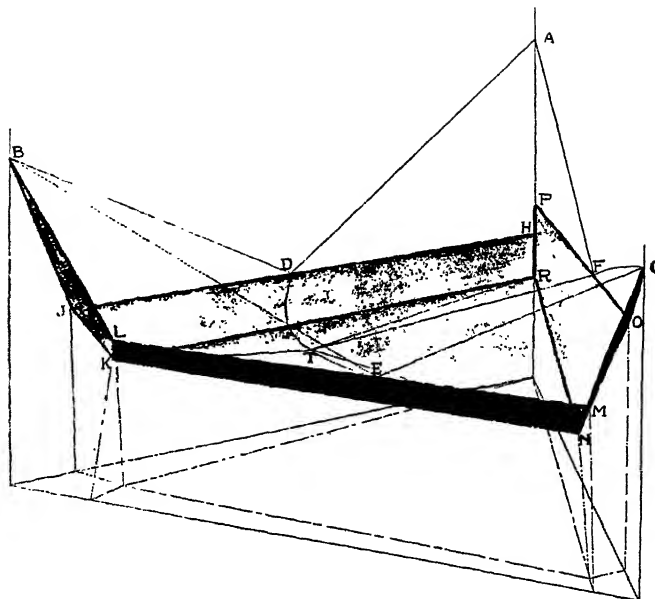


FIG. 32.—SHADOWGRAPH OF THE SOLIDUS SURFACE OF A TERNARY ALLOY MADE FROM THE WIRE MODEL (AFTER W. ROSENHAIN⁴²).

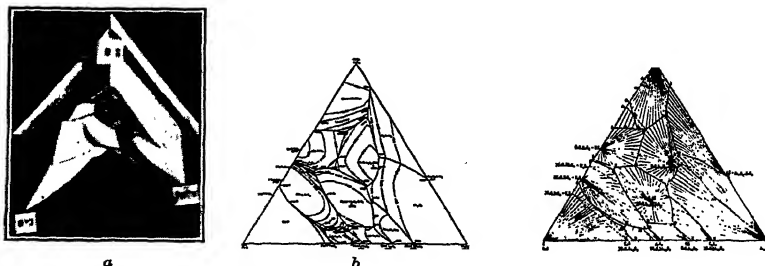


FIG. 33.—THREE METHODS OF ILLUSTRATING THE LIQUIDUS OF A SERIES OF TERNARY SOLUTIONS. *a*, SPACE MODEL OF THE SYSTEM $\text{CaO}-\text{Al}_2\text{O}_3-\text{SiO}_2$, TEMPERATURE INDICATED VERTICALLY; *b*, PROJECTION OF THE CONCENTRATION-TEMPERATURE DIAGRAM OF THE TERNARY SYSTEM $\text{CaO}-\text{Al}_2\text{O}_3-\text{SiO}_2$ WITH ISOTHERMS, MELTING TEMPERATURES OF COMPOUNDS AND INVARIANT POINTS INDICATED; *c*, EQUILIBRIUM DIAGRAM TO SHOW THE COURSE OF CRYSTALLIZATION, WHEN TERNARY SOLUTIONS OF $\text{CaO}-\text{Al}_2\text{O}_3-\text{SiO}_2$ ARE COOLED (AFTER E. W. WASHBURN⁴³).

⁴² W. Rosenhain: *Op. cit.*, Fig. 4, 253.

⁴³ E. W. Washburn: *Principles of Physical Chemistry*. 2d ed. (New York), 1921. See pp. 417-419, equilibrium diagrams for $\text{CaO}-\text{Al}_2\text{O}_3-\text{SiO}_2$.

QUARTERNARY AND QUINARY STEELS

With so complex a method necessary for the demonstration of ternary steels, it is obvious that the problem is well-nigh insoluble when we come to quarternary and quinary steels. However, various methods have been proposed and the possibilities of the solution are not quite

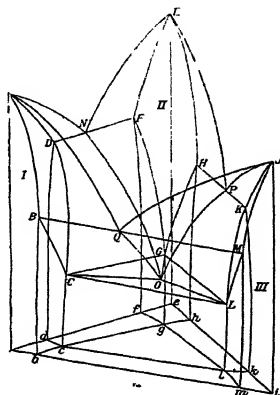


FIG. 34.

FIG. 34.—WIRE MODEL OF A TERNARY STEEL.

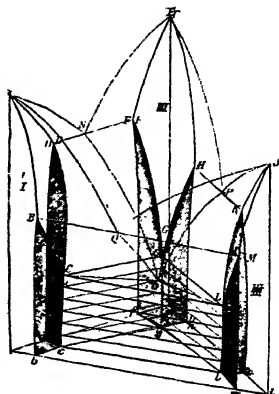


FIG. 35.

FIG. 35.—WIRE MODEL OF A TERNARY ALLOY.

hopeless. But, for the present, it seems as if the ternary steels present a problem important enough and difficult enough to challenge the skill of Howe himself. I am sure all metallurgists join with me in wishing that he were here to lead in the organization of the research and its interpretation.

Growth of Metallic Crystals

BY CECIL H. DESCH,* SHEFFIELD, ENGLAND

(Annual Lecture, Institute of Metals Division)

THE progress of metallurgical practice and the demands made by the engineering industry on our foundries and mills have made the crystalline structure of metals a subject of far more than academic interest. Metals and alloys are made up of crystals. Even if we accept the view, so ably maintained by some metallurgists, that the crystals are separated from one another by an amorphous layer, it remains true that all but a very minute fraction of any metallic mass is crystalline. The conditions which affect the growth of crystals are therefore of importance in metallurgy.

It is not only castings whose properties depend on the arrangement of their crystals, for the more we learn of the behavior of forged, rolled and drawn metals, the more certain it becomes that it is profoundly influenced by the size and arrangement of the crystals in the original ingot, determining a structure which may persist in a modified form through a long series of mechanical operations and annealings. This is mainly due to the concentration of much of the impurities in the last portions of the liquid to freeze, so that the boundaries between single crystals or groups of crystals are marked out by a zone of segregated material, differing in properties from the mass, and offering a barrier to diffusion and equalization of composition throughout the whole. The importance of this fact has been fully recognized by steel manufacturers, who have to deal with ingots which are sometimes very large, and are cast at a high temperature, containing impurities which have a marked effect on the properties even when present in very small quantity. For this reason the Iron and Steel Institute has set up a committee to examine the whole question of the heterogeneity of steel ingots,^{(1)†} and a report has been issued, containing an account of the structure of 16 ingots, ranging in weight from 15 cwt. to 172 tons, each of which was sectioned and examined by chemical analysis and by etching. A valuable body of data concerning segregation has thus been collected.

The work of the committee is being prosecuted with great activity, and at the same time researches are in progress to determine the physical

* Professor of metallurgy, University of Sheffield.

† Numbers in parentheses refer to list of references following lecture.

constants of steel within the freezing range of temperature and to examine the effect of each factor separately. Such work, which affords an excellent field for international cooperation, is of great scientific value, and should prove helpful to the manufacturer of non-ferrous metals as well as to the steel maker, the principal factors being common to both classes of metal. In this lecture I wish to consider only a part of the ingot problem, namely, the growth of individual crystals, of which a mass of cast metal is built up. Each crystal of an ingot or casting starts from a separate nucleus, and it is impossible to appreciate the structure of the mass as a whole without some attention to the units which compose it.

OBSERVATION OF GROWTH

The actual process of growth of crystals in a molten mass of metal is not easily observed, on account of the fact that all metals are opaque. At most we can watch the formation of crystals at the surface, either by cooling a large mass of metal very slowly, as has been done with crucible steel⁽²⁾ and is more easily performed with antimony or tin,⁽³⁾ or by examining a metal or alloy of low melting point by means of the microscope, using a special heating stage. Such observations give us useful information, but of a limited kind. Most of what we know of the crystal structure of metals has been gained by examining the mass after it has solidified, by cutting a section and etching it by means of a reagent, when differences of chemical composition and of orientation produce a pattern which we interpret according to the now well-established principles of metallography. To a certain extent we can trace out the manner of growth of the crystals by observing the position of boundaries, of blowholes, and of segregated impurities, but we gain less information as to the growth of a homogeneous crystal.

We are forced back on analogy. The growth of crystals is more easily seen in transparent substances, as in salts crystallizing from their solution in water, or in organic substances crystallizing from a fused mass, the former having decided advantages for studies of this kind, on account of the accuracy with which the development of the faces of crystals may be followed quantitatively by means of the reflecting goniometer, whilst changes of concentration of the solution in different regions may be determined by measuring the refractive index, the relation of which to the concentration has been established beforehand. This has been done for several salts by Miers.⁽⁴⁾

A solution must be supersaturated to a certain degree before crystallization can set in, and as dissolved salt passes out of solution and attaches itself to the crystal more salt must be supplied to it by diffusion from a distance. A solid separating from a molten mass of the same composition

affords a close parallel. Undercooling below the freezing point takes the place of supersaturation, and a flow of heat from the hotter to the cooler parts replaces diffusion. The geometrical laws are the same for the two cases, and it therefore seems reasonable to draw inferences from the one to the other. When the crystals have a different composition from the molten mass, the conditions of the two cases are combined, both supersaturation and undercooling being involved. This third class includes the metallic alloys.

ARRANGEMENT OF ULTIMATE PARTICLES

Ever since the simple geometrical relations of crystals were discovered, it has been recognized that their properties are best explained by assuming that their ultimate particles are arranged on a space lattice, and that the characteristics of that lattice determine the shape of the crystal and its physical and mechanical properties. Modern examination by means of X-rays has widened that conception and given it greater definiteness, and the atoms are now regarded as the particles forming the space lattice. The splendid achievements of investigators in this field have made it possible to state, for a large number of substances, the form and dimensions of the lattice. The characteristic of a lattice is that a unit of simple geometrical form is repeated at regular intervals in three dimensions so as to fill the entire crystal. It follows that there must be certain planes in which the atoms are more closely packed than in planes drawn at any angle to them, and it is these planes of closest packing which play the greatest part in determining the growth and other properties of the crystal.

STAGES OF CRYSTALLIZATION

There are actually two stages in crystallization, whatever be the medium in which it occurs. The first is the appearance of crystal nuclei; the second the growth of those nuclei to form crystals of appreciable size. The first is clearly a kinetic phenomenon. In the molten mass, solution, or vapor, the molecules are in a state of rapid motion and frequent collision. When the temperature has fallen so far that it is possible for a crystal to exist, a few colliding molecules may fail to rebound, and so become attached to one another, the balance between the attracting and repelling forces between them causing them to arrange themselves on a space lattice, to which other molecules subsequently attach themselves. It was once maintained that the X-ray evidence demanded that the molecule should disappear in the solid crystal, but this was a wrong inference, and in many crystals, but not in all, the chemical molecule retains its identity in the solid state. In pure metals

the atom and the molecule may be considered as identical, so that the conditions are simple.

ORIGIN AND DISTRIBUTION OF NUCLEI

How many nuclei will be formed in a given mass is evidently a problem in probability. Experiments to determine the relation between the temperature and the number of nuclei formed in a given time have been made with some success by using viscous organic substances,⁽⁵⁾ but metals offer much greater difficulties, and the few determinations which have been published are vitiated by serious errors. However, the examination of sections of ingots of the same composition and cast at the same temperature in similar molds will show that the number of crystals does not vary widely.

The distribution of the nuclei is not entirely a random one. The first nuclei originate in contact with the walls of the mold, since it is here that the temperature first falls low enough, and only the two-dimensional distribution is a random one. The crystals here are of the columnar type. In the midst of the liquid, where the equiaxed crystals are formed, there is approximately equal spacing, so that the size of the resulting crystals does not vary very widely about a mean value. If the distribution of the nuclei were entirely random there would be a much wider range, and it would seem to follow that there must be a certain periodicity of recurrence of the conditions under which a nucleus can form. We meet with similar conditions of periodicity elsewhere, in connection with the formation of dendrites, and a hint of a possible explanation may be thrown out, although experimental confirmation is needed.

The formation of a crystal is accompanied by a rise of temperature, the latent heat of solidification being liberated. Very small crystals, representing the earliest stages of growth, dissolve at a lower temperature than those of rather larger size, and we may imagine that, as the nuclei grow, the heat evolved by the larger crystals may raise the temperature so much as to redissolve any very small crystals in their immediate neighborhood. This would bring about a more even distribution, and it is conceivable that it would lead to nearly equal spacing of the surviving nuclei. An examination of a section through a mass of cast metal which has cooled with as little disturbance as possible will show that the distribution is in fact fairly uniform, so that the crystal grains are of approximately similar size and shape.

RATE OF GROWTH OF CRYSTALS

The rate of growth of crystals lends itself better to experiment and also to calculation than the rate of formation of nuclei. It is not a constant for a given substance under given conditions of temperature, etc., but varies enormously with the direction within the crystal. The

experiments of Tammann and his pupils to determine the "velocity of crystallization" by measuring the rate of growth of columnar crystals in narrow tubes, interesting as they are, give only mean values, no account being taken of the orientation of the crystals. The true velocities for different directions in salts have, however, been determined in several cases,⁽⁶⁾ and the problem is also susceptible of full theoretical treatment. We owe the theory principally to Johnsen,⁽⁷⁾ Valetton,⁽⁸⁾ and Gross.⁽⁹⁾

A two-dimensional diagram of a simple space lattice is given in Fig. 1. The atoms are represented by circles, which merely indicate their relative positions and not their dimensions. A densely packed plane is shown by the black circles, whereas the other lines drawn through the diagram are successively more oblique and correspond with faces of higher index, in which the spacing of the atoms is more open. It has been established that growth perpendicularly to a densely packed plane is

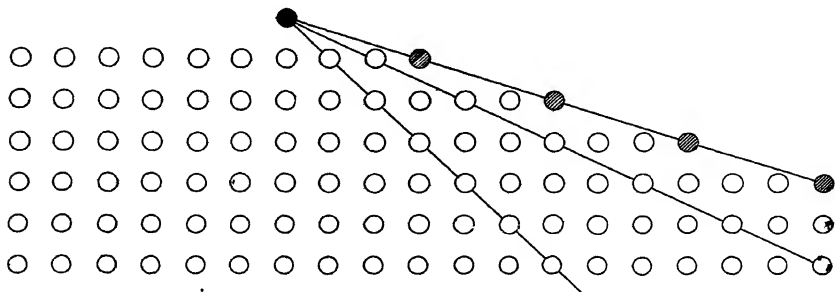


FIG. 1.—TWO-DIMENSIONAL DIAGRAM OF A SIMPLE SPACE LATTICE.

always slower than growth in any other direction. A very oblique plane is a surface of rapid growth.

It was Bravais who first showed the importance for crystal structure of the closely packed planes or, as they are technically called, the "planes of greatest reticular density." For the sake of simplicity, the diagram has been drawn for a simple cubic lattice, which is not found in metals. Its atoms are most densely packed in the cube planes, the symbol for which is (100). This position in face-centered cubic crystals is taken by the octahedral planes, (111), and in body-centered cubic crystals by the dodecahedral planes, (110). The order of density in other important planes has been calculated by Fedorow⁽¹⁰⁾ to be:

Simple cubic lattice (not known in metals): (100), (110), (111), (210), (211), (221), (310), (311).

Face-centered cubic lattice (gold, copper, silver, aluminum, lead, γ -iron, etc.): (111), (100), (110), (311), (331), (210), (211), (511).

Body-centered cubic lattice (α -iron, tungsten, molybdenum): (110), (100), (211), (310), (111), (321), (411), (210).

In hexagonal crystals, such as zinc, the basal plane, (0001), is the most densely packed.

Conditions Governing Rate of Growth

Broadly speaking, but subject to a qualification to be mentioned later, the velocity of growth perpendicularly to any face increases in the order given above. For most ductile metals, the lowest velocity corresponds with the octahedral face, (111). If the velocities in all directions could be determined and represented by vectors starting from the center of the crystal, the bounding surface would show strongly marked minima corresponding with the smallest velocities. A diagram to illustrate this

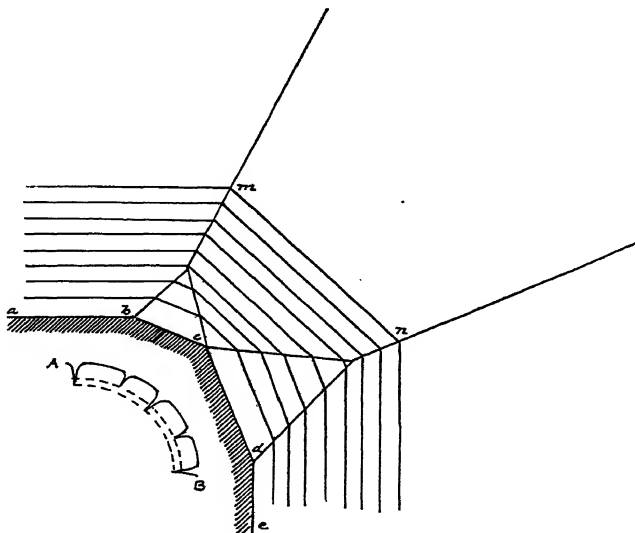


FIG. 2.—DIAGRAM ILLUSTRATING VELOCITY OF GROWTH OF A CRYSTAL.

is given in Fig. 2. The shaded boundary is that of a crystal, but the faces are not necessarily possible faces of the crystal; they may equally well be arbitrary faces prepared by grinding. The curve *AB* represents the velocity of growth, the minima corresponding with densely packed planes. It will be seen that the faces perpendicular to directions of minimum velocity increase in size at the expense of the others, and that a face *mn* which was not represented on the original crystal makes its appearance in the course of growth, and invades and finally destroys the two neighboring faces. The conclusion, which has been amply confirmed by experiment, is that the stable faces are those of greatest reticular density (Bravais' law) and of slowest growth in a direction normal to themselves.

Presence of Foreign Atoms, Ions or Molecules

Interpreted, this means that new atoms are most easily attached to a plane of low reticular density, such as is shown on the right of Fig. 1, whereas there is considerable reluctance on the part of the plane indicated by black circles to attract and hold new atoms when they collide with it in the course of their movement in the medium. This result may be considered as purely mechanical, or may be expressed in terms of chemical ideas. The forces which hold the atoms together in a space lattice are in all probability identical with chemical forces, and depend on the electronic arrangements. Within the crystal these forces are

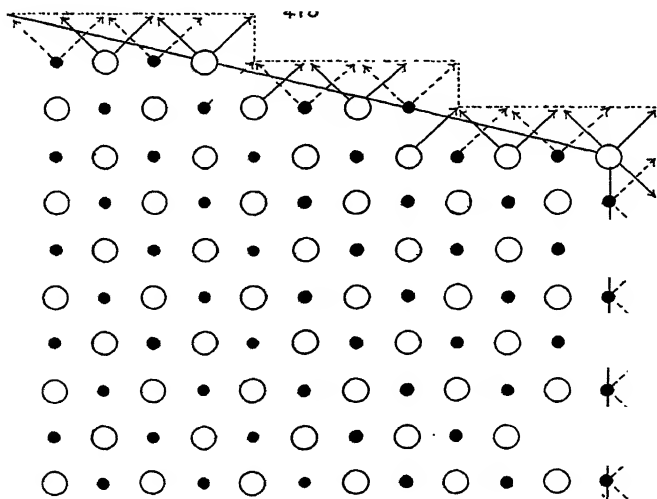


FIG. 3.—MODIFICATION OF CRYSTAL SURFACE BY PRESENCE OF FOREIGN ATOMS, IONS, OR MOLECULES.

symmetrical, but at a free surface they must have an unbalanced character, the result being the existence of "residual affinity" at the surface, which therefore has the power of attracting and holding other atoms or molecules. The holding of similar atoms constitutes growth, but foreign atoms, ions, or molecules may also be held, thus modifying the character of the surface of the crystal.

Niggli⁽¹¹⁾ has pointed out that the order of relative residual affinities may not be quite the same as that of the reticular densities, although the most closely packed planes are not affected. Assuming that the force of attraction does not extend appreciably farther than the distance between two neighboring atoms in the lattice, such diagrams as those given in Fig. 3 may be constructed, in which a face-centered cubic lattice,

that of gold for instance, is assumed, the atoms in one plane being represented by black, and those in the next parallel plane by open circles. The boundary of the region within which the residual affinity is appreciable is indicated. For the most important planes, the octahedron, cube and dodecahedron, this surface coincides with an atomic plane of the lattice, but for planes of higher index that is not so, and the boundary is broken and irregular, the velocity of growth depending rather on the extent of the irregularity than on the denseness of packing.

SURFACE TENSION

The existence of unbalanced forces at the surface must have a further effect. It must produce a surface tension, tending to reduce the surface to a minimum. There is at present no satisfactory means of measuring the surface tension of a solid, but there is much evidence that it has practical importance.

The writer has previously endeavored to show, from a statistical comparison of the shapes of crystal grains in cast metals and those of foam cells,⁽¹²⁾ ⁽¹³⁾ that the form of the boundaries is determined by the interfacial tension between the growing crystals and the liquid metal. There is direct evidence that surface tension may affect the form of a mass of metal. If we take a small mass of gold (in order to avoid complications due to films of oxide), and allow it to cool slowly after fusion, we may obtain a bead, several grams in weight, which has crystallized from a single nucleus, and is therefore a single crystal. The free surface of this is wrinkled and glassy like that of an amorphous substance. On etching with aqua regia this outer layer is removed, and the surface becomes covered with minute octahedral etch-figures. When the bead is heated at about 700° C., more than 300° below its melting point, these figures lose their sharpness, become rounded, and ultimately flow down to a level, restoring the glassy surface which was there before etching.

In the same way, minute gold crystals prepared by chemical precipitation lose their sharp outlines on heating to the same temperature, and become rounded and finally spherical. It is known that both cohesion and surface tension diminish as the temperature is raised, and it seems fair to assume that the two properties do not decline at the same rate, so that the influence of surface tension becomes greater at high temperatures, when the cohesion is relatively weaker.

Liquid Metals

It is probably the same cause which produces the rounding of dendritic crystals in alloys. As a first guess, it may perhaps be supposed that the metals with the greatest surface tension in the liquid state will also have the greatest solid tension during growth, and will thus be

most likely to form rounded dendrites. That there is some foundation for this assumption is shown in Table 1, in which the determinations of surface tension of liquid metals are collected. Unfortunately, the data here presented are not of equal value. The values in the first column have been obtained by the bubble method, which is free from objection,^{(14), (15)} and refer to the metal when just molten, whereas those in the second column were determined by the capillary depression in graphite tubes, the angle of contact being unknown, making the figures somewhat low.⁽¹⁶⁾ Moreover, these figures refer to temperatures considerably above the melting point of the more fusible metals.

TABLE 1.—*Surface Tension of Liquid Metals*

Metal	Temperature, Deg. C.	Surface Tension Dynes per Cm.	Temperature, Deg. C.	Surface Tension Dynes per Cm.
Antimony.....			840 to 850	274
Bismuth.....	269	378	700 to 850	346
Lead.....	327	452	770 to 780	424
Mercury.....	20	465	15 to 17	447.5
Tin.....	232	562	770 to 780	480
Aluminum.....			780 to 820	520
Cadmium.....	320	630		
Zinc.....	410	758	580 to 630	707
Silver.....			980 to 1120	858
Gold.....			1120	1018
Copper.....			1150	1178

In spite of the imperfections of the table, it serves to bring out the great differences between the values of the surface tension for different metals. An examination of microsections of alloys from which the metals have separated as primary crystals leads to the conclusion that there is a rough parallelism between the values of the surface tension and the tendency to form greatly rounded crystals. Slides are exhibited to show that antimony, at one extreme, forms dendritic crystals with very sharp angles, whereas rounding of the crystals is particularly well marked in silver, gold and copper. There are some apparent exceptions, for instance, lead crystals are often rounded, but this may be accounted for by the extreme softness of the metal. It is not the absolute value of the surface tension which is of importance in this connection, but its ratio to the cohesion. Crystals which are formed at very low temperatures are likely to have sharp angles. Intermetallic compounds and compounds of metals with non-metals may have low values of surface tension, as illustrated by the remarkably sharp crystals of CuAl_2 , ZnSb , FeSi , Fe_3P , etc.

FORMATION OF METALLIC CRYSTALS FROM VAPOR

A metallic crystal grows under the simplest possible conditions when it is formed from vapor. This does not correspond with any technical process at present in use, since zinc and mercury, which are prepared by distillation, are condensed to liquids and not to solids. However, the decomposition of the vapor of a chloride or other volatile compound by contact with a heated filament has been found a convenient way of preparing ductile metals, such as zirconium,⁽¹⁷⁾ which are brittle when obtained by any of the ordinary processes, and it is possible and even likely that this method will develop into a practical manufacturing process. For present purposes we may simplify the problem still further by supposing other gases to be absent, the metal being distilled in a high vacuum. When the molecules of a metal vapor, such as zinc or mercury, strike a cold surface of glass they condense, and although, according to Langmuir,⁽¹⁸⁾ most of them escape again, a metallic film is gradually built up. At first this film is structureless; that is, it consists of a single layer of molecules in a disorderly arrangement. These molecules are in movement, and as they come within the range of mutual attraction, they may unite to form a geometrical pattern, a minute, two-dimensional crystal nucleus. A surface of metal, even of this thickness, retains new molecules more readily than does the glass, so that the layer increases in thickness, and the two-dimensional nuclei grow into definite crystals. At first these are exceedingly minute, and are probably in Brownian motion, so that they will not all present the same face to the oncoming stream of molecules.

Experiments with Zinc, Cadmium and Mercury

In a most interesting series of experiments, Volmer,^{(19), (20), (21)} has allowed a narrow stream of molecules of zinc, cadmium or mercury to impinge on a glass plate which is inclined to their path, and is kept at a temperature below the melting point of the metal. Those nuclei which present their faces of slowest growth, that is, their most-densely packed planes, to the direction of the stream, have the greatest opportunity of development. For these three metals, which crystallize in the hexagonal system, it is the basal plane which has this property, and the crystals therefore grow as large, thin leaflets, having a thickness which is only a minute fraction of their breadth. When such an aggregate of thin leaflets is examined, it is found that the axes perpendicular to the basal planes all lie within a comparatively narrow cone, the central line of which is the direction of the stream. The molecules attach themselves irregularly, forming what physical chemists call an adsorbed layer on the surface of the crystal, and they then move to take up positions on the space lattice, so that the crystal grows. This supposition, that the separate molecules

do not at once enter the lattice, but that a single irregular layer is formed, within which rearrangement to a definite lattice takes place, is confirmed by observations of growing crystals under many different conditions.

Discontinuous Crystalline Growth

Electrolytically-deposited metals, lead iodide precipitated from solution, and *p*-toluidine crystallizing from solution in alcohol, have been observed to grow discontinuously by the addition of thin sheets, which there is good reason to believe are not more than two molecules thick, and probably only one molecule.⁽²²⁾ Growth is thus a discontinuous process, a layer being first adsorbed, which covers the face of the crystal before assuming a geometrical arrangement in conformity with the space lattice on which it is held. This conception makes it easier to picture the growth of crystals generally.

Some growth occurs even on those planes which have the lowest velocity, so that, in the experiments with zinc and mercury mentioned above, the leaflets increase in thickness, although less noticeably than in breadth. When so much metal has condensed that the deposit consists of an aggregate of crystals in contact, further growth can only be in thickness. The crystals now touch one another along boundaries which are not crystal planes, but are due to mutual interference, resembling in every way the grains in cast metal, although in a single layer. The free faces are often curved, as in the specimen of magnesium, prepared by distillation in vacuo, shown in the slide. The vapor escaped under very low pressure through a number of small openings in the cover of a crucible, and condensed above it. Many rhombohedral faces may be distinguished, mixed with strongly curved faces. The results obtained in Volmer's experiments are not to be expected here, as the streams of vapor diverged on issuing from the openings, and microscopical examination shows that small radiating or spherulitic groups were first formed, on which the larger crystals were built up.

EFFECT ON ORIENTATION OF MOLECULES BY DEPOSITION ON SINGLE CRYSTAL

When a single crystal of some metal is used as the surface on which to deposit, the orientation of the molecules which reach that surface is determined by it. A single crystal wire of tungsten, prepared by either of the available processes, may be heated in a mixture of tungsten hexachloride vapor and hydrogen. Should the partial pressure of the metal in the vapor be large, the atoms unite to form groups which serve as independent nuclei, and a mass of small crystals is built up, but when the partial pressure is kept sufficiently low the molecules are deposited regularly, so that the crystal grows, retaining the same orientation

throughout.⁽²³⁾ Even when the tungsten in the vapor is replaced by molybdenum, which has the same crystalline structure, the regularity of deposition persists, so that a single crystal of molybdenum, having a core of tungsten in a parallel position, is obtained, and this process may be repeated, giving a crystal composed of alternating layers in parallel arrangement.⁽²⁴⁾ Under such conditions, also, the faces may be slightly curved. The experiment of directing a narrow stream of metallic molecules against a single crystal has not yet been tried, but should prove to be a simple method of studying the relative velocities of growth of different faces.

CRYSTALLIZATION OF PURE METAL FROM MOLTEN STATE

Crystallization from vapor is the simplest case which could have been considered, because it eliminates the solvent, often a cause of complication, whilst diffusion in the vapor occurs so rapidly that variations in its velocity may be neglected in comparison with the rates of growth of different faces. Moreover, changes of temperature due to liberation of heat on condensation are without influence and may be disregarded. The next example in order of simplicity is that of a pure metal crystallizing from the molten state, so that the composition of the liquid and solid phases is the same at all stages of the process.

By enclosing the metal in a tube with a tapered end, and allowing it to cool very slowly from the narrow portion onwards, several investigators^{(25), (26), (27)} have obtained single crystals, as much as 2 cm. in diameter and up to 20 cm. long, these having grown from a single nucleus. They may have any orientation, according to the chance position of the original nucleus. This method has not been used to determine the velocity of growth in different crystalline directions, although it might perhaps be modified for that purpose.

It is known that the velocity of growth of metals from the molten state is high in comparison with organic substances, and there are thus advantages in using the latter to determine the principal facts of growth. This has been done by Nacken,⁽²⁸⁾ using an organic compound of high viscosity, salol, whilst the same experimental method has been used in a qualitative way for fused salts.⁽²⁹⁾

When a metal hemisphere, provided with cooling arrangements which can be controlled, is immersed in the fused mass, which is kept at a temperature below the melting point, the influence of undercooling and of the rate of removal of the latent heat may be studied, the heat of crystallization being removed entirely by conduction through the crystal. Cooling very slowly, the surface of the solid forms a segment of a sphere or ellipsoid, the temperature at the interface between liquid and solid being the melting point. When the removal of heat is more rapid, the velocity of production of heat by solidification at one face is overtaken

by the velocity of removal of heat by conduction. This occurs first for the face with the smallest velocity of growth in a normal direction, but as the temperature gradient increases, the same condition is reached for other faces in turn. Each face as it appears is seen as a small flat area on the curved surface. The plane faces have at first circular or oval outlines, but as they increase in size they meet along boundaries which are straight in parts, and with further growth all curved portions disappear, and the hemisphere becomes a polyhedron, the faces of which are the stable faces of a single crystal of the substance. There can be little doubt that this process, which has only been directly observed in a few organic substances, is also characteristic of metals, the thermal conductivity of which, however, is high, so that the heat liberated during crystallization is readily removed.

CRYSTALLIZATION OF AN INGOT

In the crystallization of an ingot or casting, the first nuclei make their appearance on the walls of the mold, where heat is removed most rapidly. The case of a rectangular ingot may be taken for the sake of simplicity. These nuclei are probably very numerous, but comparatively few of them grow to such a size as to penetrate far into the mass. When the etched section of a steel ingot is carefully examined, there is seen to be a thin skin of extremely minute crystals, followed immediately by a zone of large columnar crystals, approximately at right angles to the surface. This columnar zone is characteristic of ingots, and varies in depth with the temperature of casting, being as a rule most conspicuous when the casting temperature has been high. In "scorched" crucible steel ingots it may extend to the center. The change from the confused outer layer of minute crystals to the columnar zone may be regarded as due to natural selection. The original nuclei may have any orientation, but only those which have a direction of rapid growth perpendicular, or nearly so, to the wall survive, all others being quickly hidden by their more favorably situated companions.

Modes of Crystallization

In most ingots and castings the columnar zone does not reach to the center, and the remainder of the mass solidifies in crystals which have a random orientation, and are not elongated to any marked extent in any particular direction. They are said to be "equiaxed." More than one explanation of this change has been offered, but it seems most reasonable to suppose that the columnar crystals have separated under "metastable" and the equiaxed crystals under "labile" conditions. The former have grown from the nuclei in contact with the cool walls, but a point has been reached at which the temperature has fallen so

far below the melting point that the metal has entered the labile region, in which fresh nuclei can appear spontaneously.

The two modes of crystallization have been studied by Miers,⁽⁴⁾ and it is known that there is usually a distinct difference of habit between crystals which have separated under the two conditions respectively. The formation of nuclei depends on the chance meeting of molecules in the liquid in such a way that they can unite in geometrical arrangement, as a fragment of a space lattice. There is evidence to show that this production of nuclei is assisted by the existence of groups of molecules, already attached to one another, in the liquid even above the melting point. It is known that when an undercooled liquid, such as salol, is repeatedly allowed to crystallize and again melted, the number of nuclei which appear is smaller, the higher the temperature to which the substance had been heated, however slowly it may have been cooled afterwards.^{(5), (30)} This fact suggests that the high temperature breaks up the molecular aggregates which most readily form crystal nuclei. Similar influences of the maximum temperature reached while the metal was molten on the size of the crystal in ingots and castings have often been suspected in metallurgical work, and the idea is familiar, for instance, in relation to cast iron.⁽³¹⁾

The fact that the crystals of an alloy of copper and aluminum containing 10 per cent. of aluminum become larger on each successive remelting has been observed by Carpenter and Edwards.⁽³²⁾ The writer has confirmed the observation, but is not sure that the gradual removal of minute films of alumina, which are notoriously difficult to eliminate from this alloy, may not be responsible for the change. Such films can be detected under the microscope, and their gradual elimination would remove an obstacle to the growth of the crystals. At the same time, the diminution of the number of nuclei is a quite possible cause.

Manner of Crystalline Growth

Provided that the feeding of the ingot or casting is sufficient, so that there are no cavities left which cannot be filled with liquid, the crystals of both kinds grow until they meet their neighbors, the boundaries then having a polyhedral form which, as mentioned above, approaches that of foam cells. But their growth up to this point has not followed quite so simple a course as the growth of the polyhedron in Nacken's experiments, or of the single crystals in molten metals confined in tubes, or that of a salt immersed in a large mass of solution. In all these cases, a fairly simple external form is retained all through. Old faces may disappear and new faces be born, but the crystal changes during growth more in volume than in shape. This is not so in solidifying metals, which almost invariably form skeletons or dendrites, highly branched

structures, the gaps in which are filled up as solidification proceeds, until the whole mass is solid.

Among nonmetallic substances, dendritic growth is usually caused by viscosity of the liquid or by exceptionally rapid crystallization. A salt which normally forms regular crystals will separate as dendrites if the solution be thickened by gum or if it be evaporated very quickly. It cannot be viscosity which is responsible for the dendritic habit of metallic crystals, for the viscosity of molten metals is low, even near to their melting point. A much less rapid rate of cooling, however, is necessary to produce it in metals than in salts or organic compounds. The relatively high heat of crystallization of metals is no doubt a factor. The formation of dendrites may be explained by the facts of cooling, the same considerations applying to the growth of a salt in a solution and that of a metal in a molten mass.

FORMATION OF CRYSTALS AND THE THEORY OF DIFFUSION

Around a small crystal in the midst of the liquid we may imagine a sphere to be described, with such a radius that its surface may be supposed to be unaffected by the presence of the crystal. This sphere will

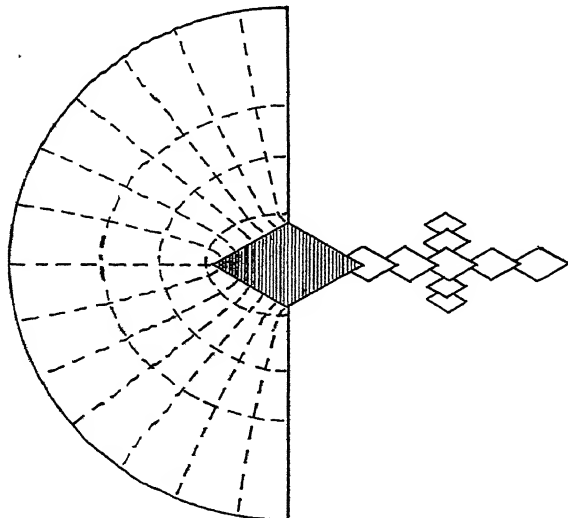


FIG. 4.—INCREASED CRYSTALLINE GROWTH AT ANGLES, DUE TO CROWDING TOGETHER OF SURFACES OF EQUAL CONCENTRATION OR UNDERCOOLING.

represent a surface of constant concentration (in the case of a solution), or constant temperature (in that of a melt). Diffusion of dissolved substance or of heat has to occur if crystallization is to continue. Within the sphere a number of surfaces of equal concentration or temperature

may be drawn, and as they approach the crystal their shape will depart further from that of a sphere, the problem of drawing them being precisely the same as that of constructing equipotential surfaces around an object in phenomena of electricity or magnetism.⁽³³⁾ Fig. 4 shows, on the left, that the surfaces are crowded most closely in the neighborhood of the sharp angles, so that diffusion is most intense there. The supply of material is therefore much more effective at the angles than in the middle of a face.

Dendrites

The mathematical treatment, based on the theory of diffusion, is not difficult,⁽³⁴⁾ but the data for calculating the result for an actual metal are not known. When growth is rapid, it will be seen that the supply to the faces may be insufficient, so that growth is confined to the angles. The more acute these become, the more the effect is intensified, and there is a tendency for the crystal to elongate at each angle, forming an acute star. Such a structure is likely to be unstable, and a slight disturbance will set up new crystal growth. A rapidly advancing point will enter a zone in which the supply is more favorable, and branching will occur as is shown in the right-hand half of the figure. The point behaves as a nucleus, with its orientation determined by the existing crystal, which extends by a parallel series of new growths, building up the dendrite. Secondary and tertiary branches are formed in the same way, and in course of time more solid is added to each of the crystal surfaces thus produced, until the gaps are filled and the crystal has become a homogeneous mass; the structure of which is only revealed on subsequent etching.

The most remarkable feature of such dendrites is their regularity. Under favorable conditions the branching takes place at regular intervals, although the perfection of snowflake crystals, which are dendrites with hexagonal symmetry, is never approached by metals. It must be supposed that the crystal exhausts the material in the neighborhood and that an appreciable time is needed for the conditions of growth to be re-established, this recurring at regular intervals. An analogy may perhaps be found in the growth of crystals in a viscous liquid with a free surface. A thin layer of salol provides an excellent example. Growth here comes to a temporary stop owing to the layer of liquid retreating from the growing tip. After a short time, it again flows to meet the crystal, and a further growth occurs, so that crystallization proceeds by intermittent steps, and the crystal is marked with transverse bands recording the successive steps, the regularity of which is remarkable.

In an ingot or casting there is no free surface, and viscosity is not the responsible factor, but the liberation and removal of heat evidently alternate in such a way as to produce a similar effect. This is one of many

periodic phenomena which are encountered in chemistry, some of which have been summarized recently.⁽³⁵⁾ The formation of Liesegang rings in precipitates is the most familiar example, while the regularly repeated twinning in crystals of potassium chlorate and in amethystine quartz is extremely striking. A mathematical treatment of the conditions for such periodic phenomena has not yet been found possible.

INCREASED GROWTH AT ANGLES—CONVECTION CURRENTS

There is another reason for increased growth at angles, besides the crowding together of the surfaces of equal concentration or undercooling, shown in Fig. 4. Convection currents are set up in the liquid, and simple geometrical considerations will show that these currents will tend to bring material to the angles. An edge has a similar effect to a solid angle, although its effects are produced in two dimensions instead of three. It thus happens that accelerated growth at the edges of crystals is sometimes seen. Crystals of native gold often have the octahedral faces bordered by a raised rim,⁽³⁶⁾ and a similar example is seen in some crystals of ferro-tungsten, in which one pair of faces of each octahedron is so bordered. The building up at the edges of the rhombohedral faces of artificial crystals of bismuth is very characteristic. A section at right angles to any edge in such cases shows that it has been formed by a process of parallel growth, the edge acting as a nucleus instead of an angle. The axes are preserved, although the external form is altered.

When the spaces between the branches of the dendritic crystals in a cast mass of metal have been completely filled up, the structure is no longer visible when a section is cut, but it may be rendered so by etching. Differences of chemical composition then become visible, but even in a pure, homogeneous metal the reagent does not dissolve a uniform layer from the surface, but its attack is localized. The direction of chemical attack in a crystal, like its mechanical properties, is determined by the position of its most closely packed planes. It must be noted that different etching reagents do not necessarily attack the crystal in the same directions, so that the distribution of chemical affinity on different planes has to be taken into account as well as the closeness of packing. The action of the reagent on a face is localized at points, which develop into negative crystals or etching-pits, the direction of which makes evident the orientation by the play of reflected light which is produced. We again meet with the remarkable phenomenon of periodicity. The etch-figures are not distributed at random, but are of nearly uniform size on any one surface, except where the metal has been distorted by strain. They are often more distinct on twinning bands than on the general surface of the crystal. To their arrangement is due the characteristic luster of an etched microsection of a metal.

EFFECT OF IMPURITIES UPON CRYSTALLINE FORM

The crystalline form of a salt may be greatly altered when the solution contains impurities. As far back as the eighteenth century it was noticed that common salt, which usually crystallizes in cubes, forms octahedra when the solution contains urea in quantity above a certain limit, and numberless other examples have been described since. The composition of the crystals is not altered appreciably. The crystals, if they take up any impurity, do so only to a minute extent, and the space lattice remains the same, the only difference being that the faces which develop during growth from pure and impure solutions are not identical.

In view of what has been said as to the residual affinity of different faces, it is reasonable to suppose that the impurity is held by the residual affinity of some faces and not of others. A layer of foreign molecules being held or "adsorbed" by certain faces, the growth perpendicular to those faces is hindered, so that they become more persistent. It is known that urea can form complex molecules with sodium chloride. For the change in habit to be due to combination would require that urea should be held more strongly by octahedral than by the cubic faces. An examination of the space lattice of sodium chloride shows that in all planes parallel with a cube face, atoms of sodium and of chlorine alternate, and are present in equal numbers. The octahedral planes, however, consist wholly of sodium atoms or wholly of chlorine atoms, the two kinds of planes following one another alternately. It is not difficult to understand that the residual affinity of such faces will retain urea molecules more readily than that of the neutral cubic faces.

This explanation is supported by the interesting observations of Gaubert⁽³⁷⁾ on the staining of crystals by dyes. Lead nitrate will crystallize in colorless octahedra, but when methylene blue is added to the solution small cube faces make their appearance on the crystals, these new faces being colored blue. As the proportion of methylene blue is increased, the blue cube faces become larger and the colorless octahedral faces smaller until, with a sufficiently strong solution of the dye, only blue cubes are obtained. It is evident that the dye has been adsorbed by the cubic planes, retarding their growth, while remaining unattracted by the octahedral planes.

Impurities in Metal Crystals

The examination of similar effects in metals is naturally more difficult, but there is good reason to believe that they occur. Impurities always, so far as is known, lessen the rate of growth of crystals,⁽⁶⁾ and therefore, other things being equal, an impure metal should give smaller crystals than a pure one. This is in accordance with experience. Alloying reduces the size of the crystal grains in an ingot or casting, and the

addition of another metal for the purpose of refining the grain is a common practice.

When a homogeneous solid solution is formed the effect may be small; thus 70:30 brass may have crystals as coarse as pure copper, the zinc atoms entering into the space lattice, but the result is different when the impurity is insoluble in the solid phase, although alloying perfectly in the liquid. The coarse crystalline grain of pure zinc is made much finer by the addition of iron, and similar examples are familiar to every metallurgist.

There are, of course, two ways in which the impurity may lessen the size of the crystal grains, one being that in question, the diminution of the rate of crystallization, whereas the other is an increase in the number of nuclei formed at a given temperature. It is not yet possible to determine the relative importance of these two factors.

There can be little doubt that the habit of metallic crystals may also change when impurities are present. One example may be mentioned. Bismuth and antimony have closely similar space lattices, but the crystals of the two metals as usually obtained are unlike one another. Bismuth forms rhombohedra which look like slightly distorted cubes, but have the peculiarity mentioned above, that excessive growth occurs at the edges of some faces, so that a stepped structure, resembling the "Greek key pattern" used in decoration, is formed. Antimony, on the other hand, forms radiating groups of hexagonal dendrites, the well known "star." In a few rare instances, masses of cast antimony have been obtained which are almost identical with crystals of bismuth, from which they are only distinguished by their color. Such a specimen, obtained in the course of manufacture, is shown in the slide. The nature of the impurity which causes such a peculiar change of habit is under investigation.

Electrolytically-deposited Metals

Electrolytically-deposited metals are convenient objects of study, as they can be kept under observation during growth by using a parallel-walled glass cell for the process of electrolysis. In the normal processes of electroplating the size of the metallic crystals is very small, as they arise from a very large number of adjacent nuclei, and the formation of crystals of appreciable size is avoided as far as possible. By varying the composition and temperature of the solution and also the current density it is possible to obtain most metals either in very minute crystals or in large dendrites. Chromium always forms so many nuclei that the resulting crystals are ultramicroscopic, which is no doubt the chief cause of the extreme hardness of electrolytic deposits of chromium. From copper, on the other hand, it is easy to obtain single crystals several centimeters in length.⁽³⁸⁾ These do not always have the same orienta-

tion. They are mostly combinations of octahedra, and are frequently twinned, giving more complex forms, and when branched crystals are found they show both octahedral and dodecahedral arrangements. The branching is most conveniently followed in lead, which readily yields (from nitrate solutions) chains of octahedra, joined as in Fig. 4, a point near to the apex of one crystal serving as a nucleus for the growth of the next, the same periodicity being noticeable as in previous examples. From an acetate solution, lead may be deposited in groups of flattened

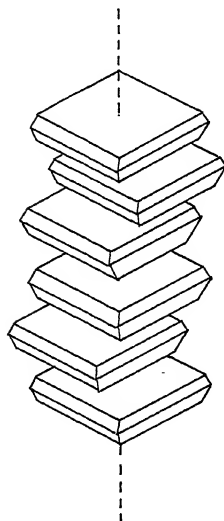


FIG. 5.—LEAD DEPOSITED IN GROUPS OF FLATTENED TABLES FROM AN ACETATE BATH.

tables, which are combinations of cubes and octahedra, as shown in Fig. 5.⁽³⁹⁾ It is probable that traces of colloidal matter formed from the organic salt are responsible for this difference of habit.

When a metal surface is clean, and a further quantity of the same metal is deposited on it by electrolysis, the crystals of the basic metal may serve as nuclei for the new deposit, so that the crystals of the latter are in parallel orientation with those which underly them. A thin film of impurity will prevent this effect, which may be seen in copper,⁽⁴⁰⁾ and even when nickel is deposited on clean copper.⁽⁴¹⁾ Phenomena of this kind are important in connection with the study of the adhesion of electrolytically-deposited metals.

The addition of small quantities of colloids to plating solutions is a common practice in industry, the usual effect being the reduction in the size of the crystals in the deposit, sometimes going so far as to produce a brilliantly smooth surface. There can be no doubt that the effect is due to adsorption, a mechanical hindrance to the growth of crystals to a large size being provided, while at the same time the growth along those axes which most readily enter into long prismatic combinations is checked. Periodic crystallization has again been observed in such cases, the potential difference varying rhythmically. There is no doubt that small quantities of organic matter are taken up in the deposit, in which they can be detected by chemical reactions, and that these adsorbed substances alter both the habit and the dimensions of the deposited crystals. On the empirical knowledge of this fact much of the success of the practical plater depends.

CRYSTALLIZATION OF ALLOYS

The object of this paper has been to show that the structure of ingots and castings cannot be understood without a knowledge of the processes

of growth of metallic crystals, and that analogies with salts and organic compounds, and with metals deposited by electrolysis, may be used to throw light on the problem. The velocities of growth are different in different directions, the governing factors being the closeness of packing of atoms in given planes, and the distribution of the residual affinities of the atoms or molecules which compose the crystal. Those affinities vary from one plane to another. It is perhaps possible to measure them quantitatively in many cases; while we may expect that, as the methods of the X-ray analysis of crystals advance so far as to enable us to determine the electronic distribution on different planes as well as that of the atoms, it may even become possible to deduce them from theoretical data. Little has been said in the paper about alloys, but it is clear that their crystallization must follow the main lines of the crystallization of pure metals, with added complications due to differences of chemical composition. Here the ordinary principles of metallography come to our aid, and there should be little difficulty in applying such knowledge as is gained in the study of pure metals to that of the more complex systems of which most of our commercial products are composed.

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Note on the Relation of Annealing Temperature to Conductivity of Copper Wire

By J. C. BRADLEY,* WATERBURY, CONN.

(New York Meeting, February, 1927)

THE relation of annealing temperature to conductivity of copper wire has been determined. Conductivity hard was 98.26 per cent. After a 10-min. heating at 200° C. it was 98.69. By annealing 10 min. at 300° C. a large increase, to 100.93, occurred. Maximum conductivity 101.15, was attained after a 500° C. anneal; thereafter it decreased to 100.53 after a 950° C. anneal. If copper is gassed the decrease is much more than this.

MATERIAL AND METHOD

The work was done on high grade electrolytic copper, 99.942 per cent copper plus silver, silver 0.0008 per cent. It was hot-rolled to $\frac{1}{4}$ in. then cold drawn to 0.080 in. The wire was cut into 25 5-ft. lengths and numbered 1 to 25 as cut. Four samples were tested hard. The others were wound into 5-in. diameter coils.

For the 100° C. heating a coil was put into water, the water brought to boiling, and the wire kept at this temperature for 10 min. For the higher temperatures the coils were placed in envelopes of sheet copper, a little water added, and the envelopes sealed by folding over the edges twice and hammering them together. This kept the air out very well. The envelope and wire were brought to the desired temperature; this was maintained for 10 min., then the whole was water-quenched.

RESULTS

Data from the tests are shown in the table and plot (see Fig. 1); conductivity is referred to the International Annealed Copper Standard at 20° C.

Conductivity hard is considerably less than conductivity soft.¹ The increase is sharp when the wire anneals. The maximum, 101.15 per cent in the particular material used, comes at 500° C.; thereafter the conductivity falls to 100.53 per cent. after 950° C. annealing, a drop of 0.62 per cent.

* Metallurgist, The American Brass Co.

¹ E. S. Bardwell: The Annealing of Cold-rolled Copper. *Trans.* (1915), 49, 755

TABLE 1.—*Annealing Temperature versus Conductivity of Copper Wire*

Sample No.	Temperature Annealed at Deg. C.	Conductivity, Per Cent.
1	20 (hard)	98.14
9	20 (hard)	98.27
17	20 (hard)	98.30
25	20 (hard)	98.31
		98.26*
11	100	98.49
2	200	98.69
12	300	100.93
13	400	101.01
14	500	101.15
4	600	101.07
6	650	100.94
15	700	100.89
16	750	100.88
22	800	100.66
5	850	100.63
7	900	100.61
8	950	100.53

* Average of the four figures above.

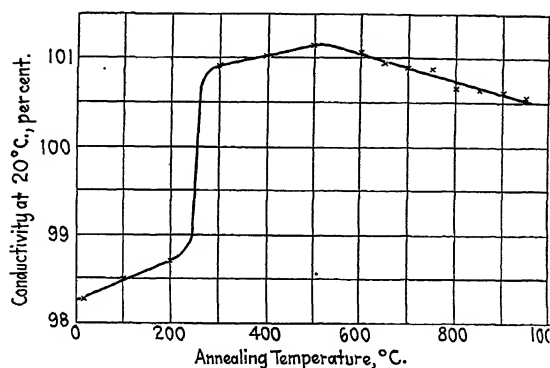


FIG. 1.—RELATION OF ANNEALING TEMPERATURE TO CONDUCTIVITY OF COPPER WIRE.

The particular interest in these figures is that similar experiments made in 1910, also showed a maximum after a 500° C. anneal, but the drop caused by annealing at 950° C. was 2.45 per cent., being much greater than found in the present investigation. The earlier material was probably gassed at the high annealing temperatures, for the heating was done in a mild steel bomb filled with steam; the steam might readily have been decomposed, forming hydrogen, which would have penetrated the

copper and caused gassing. In the present experiment the container was copper, not iron, and the absence of reducing gases was assured. Microscopical examination proved that no gassing had occurred in this instance.

DISCUSSION

COLIN G. FINK, New York, N. Y.—Referring to Curve 1 in Mr. Bradley's paper, I would like to inquire whether that break in the curve at 500° depends very much on the annealing atmosphere. Mr. Bradley referred to a steam-hydrogen atmosphere. The "steam" probably persisted during the lower temperatures of the anneal, but I do not believe very much steam was left toward the end, probably largely hydrogen and other reducing gases. In my experience if we maintain a dry hydrogen atmosphere above 500° C. we get a continued upward and not a downward slope. The conductivity increases as the annealing temperature increases beyond 500° . Likewise the temperature coefficient of resistance increases as the annealing temperature (in hydrogen) increases.

Adaptability of Various Coals as Generator Fuel in the Manufacture of Water Gas*

By W. W. ODELL,† PITTSBURGH, PA.

(New York Meeting, February, 1927)

ONCE it was believed that anthracite or coke were the only fuels generally available and suitable for the generation of water gas, particularly so when this gas was made in the generators of standard carburetted water-gas sets such as are commonly used in the manufacture of city gas. During the past 8 years, practice has changed considerably and it is now common to find gas plants operating without anthracite or coke and using bituminous coal in the generator. This change in fuel has appreciably increased the marketability and sales of certain coal producers' products; accordingly it is of interest to coal operators and others to know the reason for this change, the present trend in practice and the adaptability of various coals for generation of water gas. This paper briefly discusses these conditions and some of the factors relating thereto.

BITUMINOUS COAL NOT IDEAL BUT CHEAPER THAN COKE

In spite of many changes which have been made in both process and generating apparatus for using bituminous coal as generator fuel more efficiently, in general it is not an ideal fuel for this purpose. It is supplanting coke because it costs less; it is so much cheaper per ton delivered at the gas works that water gas made therefrom costs less per unit-volume than that made from coke, even though the quantity of generator fuel consumed per therm (100,000 B. t. u.) of gas made is greater with coal. This is the reason why so many city gas plants, as well as others, are using bituminous coal instead of coke.

CONDITIONS FAVORING CHANGE FROM COKE TO COAL

The conditions which have made it possible and feasible, as well as economical, to supplant coke with coal may be summarized as follows: (1) Changes in methods of operating and alteration in generating apparatus; (2) lowering of the calorific standards for city gas, and (3) availability

* Presented with approval of the Director, U. S. Bureau of Mines.

† Fuel Engineer, U. S. Bureau of Mines.

of clean, sized bituminous coal at a price appreciably lower than that of coke.

Condition (2) obviously applies only to city gas plants. However, considering the probable expansion of the manufacture of synthetic methanol wherein methyl alcohol is made from blue gas (water gas), and the possible use of water gas in the synthesis or hydrogenation of other products, interest centers not alone in the city gas plant but in any plant generating gas by the action of steam on incandescent carbon. "Straight water-gas" plants may be used more extensively in the future as industrial fuel in certain heating operations as petroleum products become less available or higher in price. One is interested then in the properties of coal which influence operating results in making water gas and in the results from variations of plant operating practice.

NUMERICAL VALUE OF COALS FOR GENERATOR FUEL

Efforts have been made to assign a numerical value to various coals with respect to their relative usefulness as generator fuel, but that is a difficult task, because there are so many factors to be considered. Of two coals *A* and *B*, *A* may be superior to *B* with respect to certain properties such as calorific value and ash content, but so inferior to *B* in other respects that it could not be rated any higher than the latter as generator fuel. Likewise, coals *A* and *B* may be so much alike as to be thus rated in most cases and yet under some operating conditions, or rather under the conditions prevailing in some plants, one may be ruled out as unsatisfactory and the other judged as suitable.

ADAPTABILITY OF A COAL FOR USE AS GENERATOR FUEL

Some of the important properties of coal which should be considered under this heading are: (1) Composition, with respect to moisture, volatile matter, fixed carbon, and ash; (2) calorific value; (3) size; (4) friability; (5) slacking properties; (6) fusibility of the coal ash, (7) cost per ton delivered at the plant, and (8) coking properties.

In selecting coals for generation of water gas, it is possible, by the aid of laboratory tests to rate certain coals as superior to others, but in many cases it is necessary to make actual plant tests in order to select the one most suitable. The items enumerated above and which are usually considered in evaluating coals are discussed in the same sequence.

It may not be superfluous for the better understanding of what follows to state that the water-gas process is primarily a two-stage process whereby gas is made intermittently: First, there is air-blasting of the ignited fuel in a cylindrical shell (generator) to incandescence; and second, discontinuing the blasting and introducing steam into the fuel bed, then col-

lecting the water gas generated. It should be noted that the air-blast gas is either discharged into the air, burned and the heat of combustion utilized, or passed through a waste-heat boiler where the sensible heat is utilized. The fuel requirements are, or may be, different in each method.

COMPOSITION OF COAL AS REVEALED BY PROXIMATE ANALYSIS

Moisture

The simple proximate analysis of a coal gives a clue as to what might be expected of it when used in the generator. It reveals the percentage content of moisture, volatile matter, fixed carbon, and ash, and to a certain extent the coking properties.

Other things being equal, coal with the lowest moisture content is preferable. This is true for obvious reasons and also because it is more difficult to heat and completely carbonize coals of high-moisture content, or at least a longer period is required to accomplish this. In other words, a smaller percentage of the fuel in the generator will be heated to gas-making temperatures for a given set of conditions. The natural result of this is a decrease in gas-making capacity. However, other things are never equal, and it is commonly the case that moisture content is not a serious consideration in comparison with other factors when using the better grades of coal.

Volatile Matter

The percentage of volatile matter present in a coal is, to a certain extent, a barometer to the method of operating. Conditions with respect to generating apparatus, process and gas standards have been "molded" about a low-volatile fuel such as coke or anthracite, and it is therefore not surprising that when high-volatile fuels are used in place of the cokes or anthracite a certain percentage of the volatile matter is lost in the stack gas; the loss is greater the higher the volatile content of the coal used. Clearly this means a greater quantity of such fuel must be used per 1000 cu. ft. of gas made than with coke or anthracite of the same calorific value. It has been found possible, however, to change the operating procedure in the manufacture of city gas so that less volatile matter is lost during the blasting period than formerly. The present trend toward lower gas standards has aided in the use of the bituminous coal. Under conditions prevailing in the manufacture of city gas, or of blue gas for industrial use, coals with appreciably large quantities of volatile matter can be and are satisfactorily used. It should not be inferred that a high-volatile coal is now preferable for generation of water gas, for such is not true. However, by using certain recently developed improvements in generating apparatus, there are conditions where best results can be obtained with coals of high volatile matter.

Fixed Carbon

True water gas is generated by the action of steam on carbon heated above 1000° C. (1832° F.). This is a temperature above that at which volatile matter is determined by the accepted methods of analysis, therefore the volatile matter¹ will be driven off before the true water-gas reaction starts. The actual reaction will therefore be the same as if the coke or char had been produced by a separate coking operation, instead of in the generator. This is a point sometimes forgotten in studying the possibilities of using coal as generator fuel, and it may help clarify the statement that "high-volatile coals are not ideal for making water gas."

Ash

The ash content of a coal has considerable bearing on its usefulness. The greater the amount of ash present in the coal the more frequently it will be necessary to clean the fires and remove clinker. The gas-making capacity is decreased as the period required for clinkering is increased. A further disadvantage in the use of high-ash coal is the increased amount of combustible matter in the ash removed and the rapid increase in resistance to the flow of air and steam through the fuel bed between clinker periods. The latter condition is occasioned, to a great extent, by clinkering of the ash and is not alone proportional to the ash content of the different coals, but is a function of the percentage of ash present and the type of clinker it produces in the generator. As a rule, the maximum capacity can be obtained with the fuel of lowest ash content, other factors remaining constant.

The proximate analysis of a coal therefore tells us, within certain limits: (1) How much trouble might be expected from excessive accumulation of ash and clinker; (2) to what extent it will be necessary to employ modifications of the standard methods of operating because of high-volatile matter; (3) the difficulty in heating the coal mass to a gas-making temperature; and (4) roughly, the fuel required per 1000 cu. ft. of gas made.

CALORIFIC VALUE OF COAL

In conjunction with the proximate analysis of coal, the determination of its calorific value not only helps in its classification but makes possible a closer estimate of the results which may be obtained by its use in a generator; in fact, it is essential to consider relative heating values of coals to be selectively chosen for gas-making purposes. If a coal having a calorific value of 14,000 B. t. u. a pound is used, and 40 lb. of it are required per 1000 cu. ft. of gas made, it is obvious that if a change is made to a coal of 12,000 B. t. u., more of the latter will be required in

¹ There is, of course, some volatile matter left in the coke when coal is heated to this temperature, but the percentage is small and is of no practical importance.

making the same volume of gas. Without consideration of other differences, the coal requirements with the latter fuel would be $\frac{14,000}{12,000} \times 40$, or 46.7 lb. This difference of 6.7 lb. would be less if the latter fuel had a lower volatile content than the former, and vice versa.

The true value of a lignite or lignitic coal might not be detected by one not familiar with such a fuel, by proximate analysis alone; but when the calorific value is also known the coal can be more correctly evaluated.

SIZE OF COAL FOR GENERATION OF WATER GAS

The importance of using clean sized coal, substantially free from dust and fines, in order to obtain the highest efficiency in gasification, is now recognized by gas engineers and provision is made at many gas works for removing the slack and fines before the fuel is charged into the generator. Numerous attempts have been made by different investigators to use lump coal containing appreciably large quantities of fines, but the result in each case has been that clean coal was recommended. One experienced operator, using Illinois coal, states this:

The most important factor in the successful use of bituminous coal as generator fuel is the proper sizing and separation from the fines before charging.

When slack or fine sizes of coal are introduced along with the lump coal, the tendency for channels or flues to form in the fuel bed is materially increased, and this results in the blowing over (out of the generator) of appreciable quantities of the fines. In the case where carburetted water gas is made, the checker chambers become clogged with coal and the oil-cracking efficiency decreases. There are other serious results, just as important, resulting from this practice, chief of which is decreased gasification efficiency accompanied by decreased production of gas.

In some plants provision is not made for separating the fines; the coal is purchased as sized lump coal such as large egg or 3 by 6-in. lump coal, and this is used just as received. Under these conditions one coal may be accepted, or it may be rejected in favor of another not strictly because of the nature of the fuel itself but because of a difference in operating results attributable to differences in the percentage of fines in the respective coals. A good coal improperly prepared is or may be inferior to a lower grade coal properly prepared. The writer recently visited a large gas plant in the Middle West where excellent results were being obtained and noted that in spite of careful handling of the coal, which was purchased as sized lump, provision was made at the generator for separating the fines. This simply consisted of a chute having a false bottom of closely-spaced bars or grizzly from the overhead storage hopper

to the generator. In passing over the grizzly the fines passed through and were deflected from the path of the lump coal.

FRIABILITY OF COAL

Because of the deleterious effects of the presence of fines in sized-lump generator fuel, it is obvious that a coal for this purpose should not be friable. After being loaded on cars for shipment, considerable handling and rough treatment, which most coals undergo before reaching the generator, result in appreciable quantities of fines, which increase as the friability of the coal increases. Again, when fuel is charged into the generator the drop may be such that fines are formed. Other factors remaining the same, the less friable the coal the better it is for generator fuel. This should be remembered by those connected with the preparation and shipment of coal to gas plants.

SLACKING PROPERTIES

So far as the use of Eastern coals is concerned, little need be said regarding the tendency of some coals to slack upon brief exposure to the atmosphere. With some Western coals bordering upon sub-bituminous or lignite this is a distinguishing property. Such coals are prone also to disintegrate upon being heated; hence, even though the storage period be reduced to a minimum, trouble may be anticipated. Much skill and care are required to use such a fuel and even then most gas engineers would pronounce it "an impossible fuel," but there are places in this country, particularly on the Pacific Coast, where because of difference in fuel costs, such a fuel is not "impossible." A sub-bituminous coal crushed to less than $2\frac{1}{2}$ -in. size, and ranging from about $\frac{3}{4}$ to $2\frac{1}{2}$ in., has been used for a considerable period at Tacoma, Wash. For obvious reasons such a fuel can not be recommended when a higher grade fuel is available at a satisfactory price.

ASH FUSIBILITY

With some operators it is a question whether a coal with an ash of high fusion point (say 2600° F.) is superior to one of low fusion point (say 2200° F.). It is the writer's observation that when the ash is very low there is little difference in results with coals of either extreme. The influence of the properties of a coal ash upon operating results is more pronounced as the percentage of ash increases, and under these conditions a coal with an ash of high fusion point is to be preferred. However, excellent results are being obtained with Illinois coals which give ashes of low to moderately low fusion points (say below 2200° F.). Because of differences in methods of operating and advantages obtainable with recently improved apparatus, it cannot be said that for all conditions one

coal is better than the other. But it is certain that when a coal of high ash content of low fusion point is used, the ash will rapidly accumulate as clinker in the generator and offer resistance to the passage of air and steam, thereby decreasing the capacity of the generator. It is to be noted that the advocates of the automatic and continuously operated clinkering devices claim that they prefer an ash of low fusion point because it can readily be put on the grates where it can be taken care of. Without unduly prolonged discussion of the various conditions of operation, relative values cannot be placed on these variables.

COST OF COAL DELIVERED AT THE GAS WORKS

Any coal of high-calorific value is adaptable for gas-making if the cost is low enough. In fact, bituminous coal is adaptable chiefly because it is cheaper than coke, and this differentiation can be carried clear through the various grades of coal. The most important point to the gas manufacturer is the cost of gas per therm. If gas can be made more cheaply from a coal that offers some difficulties in its use than from other coals, the former class will be used and an effort made to alleviate the difficulties. It is probably well that this is true for it tends to decrease the distant shipments of coal and gives the local coal operators a chance to supply fuel to local gas manufacturers. A current example of this is seen in the Middle States: A certain West Virginia coal has been used there with excellent results in gas generators and many of the gas manufacturers prefer this coal; however, because of a difference in the delivered cost it has been found in some places that with Illinois coal, gas can be made at a lower cost per therm, even though a little more is required per 1000 cu. ft. of carburetted gas made.

COKING PROPERTIES

The effects of the coking properties of coal on results depend upon other factors, and it is difficult to isolate the coking factor. There seems to be a difference of opinion among operators whether or not it is desirable for generator coal to have coking properties. Some maintain that when a coal has coking properties there is a tendency for the fines to coke and become cemented to adjacent particles, thus reducing the amount of "blown-over" coal. This appears to be a "belief" rather than a proven fact, because it is the writer's experience, which has been checked by others, that with coal of strong coking properties there is a tendency for it to coke in the upper portion of the fuel mass, causing arching of the fuel, the formation of flues, and aggravating the condition for the blowing over of fines. This means that it is even more essential to use clean sized coal when it has strong coking properties than when it has but little tendency to coke. It is, of course, important that the individual lumps of fuel supplied the generator retain their entity upon being heated,

but some non-coking coals tend to break up or disintegrate upon being heated. This is probably the reason why a coal with some degree of coking property is preferable.

When coal is charged into a generator the outer surfaces of the lumps are heated rather rapidly; this change is somewhat greater in winter than in summer. It should not be surprising to find that a certain amount of chipping and spalling takes place because of this rapid heating, and that such disintegration occurs below the coking or fusing temperature. If this occurs, and the writer is sure that it does occur, it would be interesting to know to what extent it takes place and how widely this property varies with different coals. So far as the writer is aware no tests have ever been made to determine the extent of this action with any coal. It is difficult, then, to assign a value to coking properties until the influence of certain other unknown factors have been determined.

From the foregoing it will appear that there are few if any coals that are 100 per cent. perfect with respect to their use as generator fuel. In selecting a coal, the best that the gas manufacturer can do is to consider the advantages and disadvantages which will result from the use of different coals and then, with consideration for differences in costs, make a choice. It is hardly possible for each gas plant to make exhaustive tests with a large number of coals under a wide variety of operating conditions, hence it appears that it devolves upon the coal producer to ascertain the conditions under which his coal will give best results and sell it with that information.

In effect, it may be said that the ideal specifications call for a hard, clean sized, non-friable, non-slacking coal low in moisture, ash, and volatile matter, with a high calorific value, whose lumps retain their shape or entity in the generator. In making carburetted water gas there are conditions when it is advantageous to have a moderately high percentage of volatile matter in the coal.

Fig. 1 shows graphically the relation between the weight in pounds of coal used per 1000 cu. ft. of blue gas made and the cost of generator fuel per M cubic feet. With this chart the relative values of two different coals can be determined,[†] other factors being the same. For example: it will be seen that 40 lb. of coal costing \$5 a ton is equivalent to 50 lb. of fuel costing \$4 a ton. Sometimes it is economical to use a coal of which more is required per M. cubic feet of water gas, than a coal costing less per ton. No corrections are made in the chart for differences in the cost of handling and storing the coals, and removing the ashes.

OPERATING FACTORS INFLUENCING RESULTS WITH COAL FUEL

Before concluding this paper it is desirable to point out that operating factors which are or may be different in the various gas plants so influence

results that a particular coal may be satisfactory in one plant under a definite set of conditions and unsatisfactory elsewhere under different conditions. Some of these are: (1) Operating cycle, including method of making up-and-down runs or split runs; (2) operation with or without the blow-run, back-run, and pier processes; (3) air-blast pressure employed and rate of gas-making per unit of operating time; (4) method of charging the generator; (5) coal-handling equipment, and (6) operation with or without a waste-heat boiler, etc.

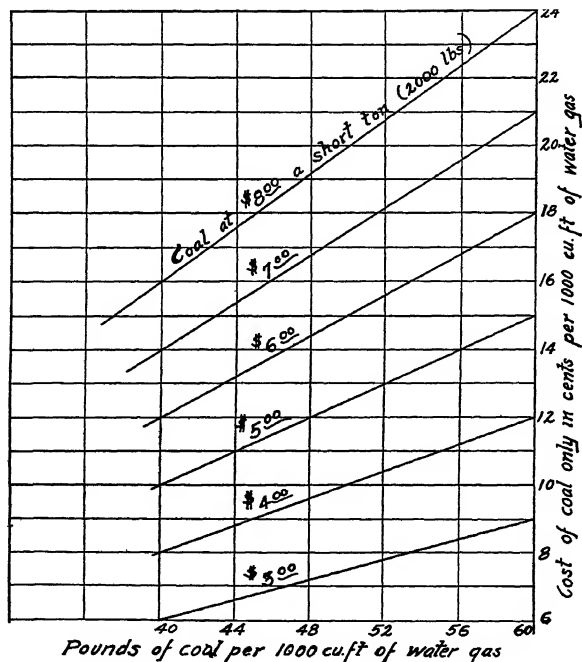


FIG. 1.—COST OF GENERATOR FUEL USING 40 TO 60 LB. OF COAL PER M. CU. FT. OF GAS, AT PRICES RANGING FROM \$3 TO \$8 A TON.

It is not within the scope of this paper to present detail methods of operation; therefore, it will suffice to say that: (1) Clinker conditions can be controlled to a considerable extent by adjustments in operation; (2) the blow-run, back-run and pier processes have made possible economies in the use of bituminous coal not previously obtained (it is reported that less channeling occurs and less of the fines are blown out of the generator when using the latter process); (3) with high-blast pressures, troubles may be experienced with blown-over fines not experienced with low-blast pressures (under 30 in. of water); (4) the use of automatic

charging or fuel-spreading devices influence the conditions in the generator fuel bed; (5) poor results in using a particular coal are sometimes attributable to a lack of facilities for properly handling the coal after delivery; (6) the use of a waste-heat boiler permits the recovery of an appreciable amount of the heat-loss which might otherwise be charged directly to the high volatile content of a coal. In other words, fuels can be used in the generator with high thermal efficiencies when a waste-heat boiler is used in connection with the gas-generating set and the heat recovered thereby is greater, as a rule, with high-volatile coal.

DISCUSSION

W. H. FULWEILER, Philadelphia, Pa.—There are several points to be considered relative to bituminous coal for gas making. What kind and quality of gas are you going to make, and what are you going to use it for? What are the quality and the cost of oil, the quality of the coal? Is there a coal gas works available? Finally, there is the price of coal.

I think Mr. Odell is a little optimistic when he expresses the belief that bituminous coal is going to be the ideal fuel for use in making blue water gas for synthetic operations. There are certain synthetic operations where they simply will not have it, because it is very desirable to have a low percentage of methane. Blue water gas made from bituminous coal contains sometimes as high as $5\frac{1}{2}$ per cent. of methane. If blue water gas is used for heating then the use of bituminous coal is entirely proper because it gives a higher heating value.

When we use the bituminous coal for city gas we have a different situation. When we are working with a lower heating value gas around 450 to 550 B. t. u. we can secure greater efficiency with the use of bituminous coal than when we are running a very high heating value gas. This is really dependent, however, upon the use of what we call the blow-run. This is an endeavor to save that portion of the blast which is high in heating value. With coke, the heating value of the blast products runs from 60 to 80 B. t. u. per cu. ft., whereas with bituminous coal it may run as high as 150 B. t. u. per cu. ft. It is very evident that if with blow-run operation we can save some of this 150 B. t. u. gas we are going to increase our efficiency.

When we begin to discuss results from the use of the blow-run with bituminous coal we must be careful because it is evident that the more of this producer gas that we put into the gas, the lower will be our fuel results. Yet, on account of its low heating value the more oil we will have to use to bring the mixture up to the standard of heating value that is required. It appears, therefore, that we must say, speaking of results with bituminous coal blow-run, both how much oil we use per 1000 cu. ft. and how much fuel we use per 1000 cu. ft., because the two are nearly interchangeable.

In other words, it is quite possible to exchange coal for oil. Where there is good coal and a reasonable oil, it is practically on the basis of 1 gal. oil for $\frac{1}{100}$ th of a ton of coal. In other words 5 c. oil is \$5 coal; 6 c. oil is \$6 coal.

When producer gas is made, it is produced quite rapidly, and by using more of it the capacity of the set can be greatly increased because the blast is put through considerably faster than the gas comes through. If the oil cost is of a certain figure it is possible to exchange oil for a combination of fuel and capacity. In other words, as the blow-run is increased the capacity is increased, but the fuel is decreased and the oil is increased. By using 0.2 gal. oil extra, capacity may be increased about 22 per cent. That brings a third situation: What is the capacity, the generating capacity of the plant compared with the output? It may very well be that we can afford to spend the extra oil to get the increased capacity and saving in fuel.

I agree with Mr. Odell's specification as to coal. However, he does not tell us how to enforce it. Suppose I want a hard coal, a non-friable coal. There are no tests at this time to tell us what is a hard coal, a non-friable coal, or coal that will not break up in the generator.

There are many different ways of handling the coal size, and it is a little hard to evaluate which coal is the most important. I think we must give a great deal of attention to this because there is no question that the size of the coal that actually gets into the generator is a determining factor.

The action in the fire is important. Some coals when you drop them into the fire go to pieces, and form large amounts of fines which are blown over into the blast. In others, the whole top will run together and fuse up solid. Other cokes will apparently fuse together, but that is only for about a run and a half, maybe two runs. Then, fissures begin to develop, and instead of forming a blow hole you really have formed a sizeable piece of coke which appears to be quite active.

How can we test a coal in the laboratory and tell how it is going to operate in the set? We are making some tests, dropping coal into little furnaces heated at different temperatures to see if we cannot get some differentiation. The difficulty is that we have not a great deal of data to correlate. We have four or five coals that we know all about, but the tests so far are so erratic that we have not gotten very far.

Relative to coals of high and low volatility: My experience to date has been that all low-volatile coals act badly in the fire. We have yet to find one that will stand up. Apparently we must have somewhere above 30 per cent. of volatile to get the best results. I have not seen successful results below that. Thirty to 35 per cent. of volatile gives a tougher piece of coal when heated.

The coking index apparently does not help us at all. One of the best coals we have has a coking index of 12, and the next best has a coking index of 19. That is a tremendous variation.

There is a decided difference in the action of the clinker. We have used the bituminous coal in the generator and then used coke made from by-product ovens, and there is no question that something goes on in the generator, a little different action, and that we can handle the clinker in probably 60 per cent. of the time with the coals than with the coke. One nice feature in using coal is that the ash is lower. It is only about 60 per cent. of what it was in the coke because you start with a volatile there.

So many factors must be taken into consideration that the final delivered price of the coal is not, after all, of the first importance. It is important however, that local coals are available which can be put in competition with the imported coals. Then, and this is even more important, by utilizing bituminous coal a competitor is established for the coke. From the gas man's standpoint it has been the finest thing in the world to be able to use this bituminous coal. It caused a drop of \$2.50 a ton in the delivered price of coke merely by ordering 10 cars of soft coal.

W. J. MURDOCK, Joliet, Ill.—We used soft coal as generator fuel 15 or 16 years ago in some of our smaller plants and with small water gas machines. During the war period in the Middle West, when it was almost impossible to get coke and anthracite coal, a great many of the small companies were forced to the use of bituminous coal as the only other available fuel, and it meant to them the difference between getting along and bankruptcy.

Along about 1921, we were forced to choose between shutting down a large coke oven plant and using 100 per cent. bituminous coal in the 11-ft. water gas machine. Many more difficulties were encountered working with large machines than had been previously encountered with coal in a small machine. In a small machine the refractory wall is nearer to every portion of the fuel bed, and that feature plays a rather important part in the successful use of soft coal as water gas generator fuel.

To begin with, when bituminous coal is used, there are four zones in the fire, where ordinarily with coke or anthracite coal there are only three zones. In other words, when coke or anthracite is used, there are the cold fuel, the incandescent zone, and the zone of ash and clinker. With soft coal, there is the cold fuel, then you have to go through the distillation zone, to the incandescent zone and the ash and clinker zone. In the use of soft coal all the difficulties are encountered in passing through the distillation zone. Soft coal will mat and form a mass that is almost impervious to blast and steam and the path of the blast and steam will pick the weakest part and pass through, causing considerable channeling and very much decreased efficiency in operation. This causes a lot of small particles of the fuel to be carried over into the checker chambers. We encountered this condition.

In our early stage of operation with the 11-ft. water gas machine, we made about 1,200,000 cu. ft. a day, and with very clean sized coal possibly 1,500,000 cubic feet.

We tried many and varied methods of trying to eliminate this difficulty. At one time we went as far as putting a solid plate with about a 20-in. hole in the center of it on top of the grates, thinking we would be able to conduct the blast and steam toward the center of the fire. The fuel in the center formed an inverted cone which was practically dead coal all the way through. We then tried mechanical agitation—top agitation with the stirrer of producer type. This filled the holes temporarily, but had a tendency to grind the fuel up and possibly make it worse, not better by any means.

We learned in the early stages that clean sizing of the fuel was essential, due to the fact that there is a segregation in the fire when the fuel is charged. No matter how clean in size the coal is, with the larger lumps falling to the side of the generator, and the smaller dropping to the center, the path of the blast and the steam is up along the outer walls.

This condition led us to the development of what is now termed the pier process, which consists of a refractory cone or pier interjected in the center, and on top of the grates extending from the top of the grates to about level with the off-take of the machine. In putting this into a machine we take out about 10 per cent. of the original grate area, but increase the wall surface by about 33 per cent.

The pier stores heat during the blasting period, which is given up to the fuel during the gas-making period, helping and speeding up action through the distillation zone by the radiant heat given from the refractory surface. In other words, we find that if the pier is properly designed, the action around the circular pier or around the refractory pier rather, extends part way into the fire, and with the action around the outer surface extending in, there is a uniform processing of the fire all the way through.

We have been able to get practically pound for pound result with bituminous coal of a given grade as against a coke of a like grade. With a good coal lower fuel results, possibly better economies, may be obtained, but it is the last figure on the balance sheet that governs in selecting the fuel. In the Middle West there are many local coals, and it behooves us to use the coal that is the cheapest in the end. We are at the present time using 100 per cent. Illinois coal with very satisfactory results. I understand that in the far West there are some Utah coals that have characteristics of Illinois coals, and I believe on the west Coast they are planning to do some experimental work with them at the present time.

W. H. BLAUVELT, New York, N. Y.—Mr. Sweetser's propaganda for intelligent and scientific preparation of specifications for coal for various purposes, is emphasized by discussion. I have taken the position in discussing Mr. Sweetser's ideas that each branch of metallurgy, if we may include gas-making in that, requires its own specifications, and Mr. Fulweiler brings out that thought very clearly.

In the substitution of coal for coke in the making of water gas each problem must be taken up by the individual gas manufacturer, and studied under his own conditions. Mr. Murdoch had very little trouble when he used small sets, but with the large sets

he found a different set of conditions, which, by the way, apply to producer gas as well as water gas. He had to adapt his process to the larger method.

We must develop specifications for coal and coke in this field, which is a very important one and very interesting to all coal producers.

H. C. PORTER, Philadelphia, Pa. (written discussion).—As Mr. Odell points out, the relative friability of a coal is a very important factor in its adaptability to use as a generator fuel. Coal operators are preparing now 3 by 6-in. lump for this purpose. The operator's costs are greatly influenced by the amount of nut and slack that he has to find market for at prices well below those of the lump, and this proportion depends on the friability and the amount of handling.

The gas works also must handle the coal. And the generator operation and plant operating costs are affected by the fines that get into the generator, even with lump coal, by reason of this handling. L. B. Eichengreen,² in a recent article on bituminous generator operation, refers to one plant where unusually good results were being obtained, and the coal, just before entering the generator, was passed over a bar screen chute.

At the Bureau of Mines, some 14 to 15 years ago, I made comparative friability tests³ on a number of coals, using a drop test in which a 50-lb. sample of screened 2-in. lump was dropped four times from a height of 6 ft. onto an iron plate, and the fines determined by screening through 2, $\frac{1}{2}$ and $\frac{1}{8}$ inch.

Most of the coals, after the four drops, gave 40 to 70 per cent. passing 2 in., 6 to 30 per cent. passing $\frac{1}{2}$ in., and 2 to 13 per cent. passing $\frac{1}{8}$ inch. These are important differences. The Western coals (with the exception of one Washington sample) showed up well, and Illinois and Pittsburgh samples were of considerably lesser friability than those samples of low-volatile smokeless coals of Pennsylvania, West Virginia, and Virginia that were tested. The following were the results:

COMPARATIVE FRIABILITY OF CERTAIN COALS
[2-in. Screened Lump, after Drop Test]

Coal	Held on 2 in., Per Cent.	Through 2 in., Per Cent.	Through $\frac{1}{2}$ in., Per Cent.	Through $\frac{1}{8}$ in., Per Cent.
Clearfield District, Pa.....	30.0	70.0	30.5	13.0
Fayette Co., W. Va.....	30.8	69.2	27.0	11.5
Pocahontas, Va.....	45.5	54.5	25.0	11.6
Navy Coal, U. S. S. "Maine"*.....	30.5	69.5	22.2	8.3
Black Diamond, Wash.....	25.4	74.6	25.4	5.4
Pittsburgh District, Pa.....	54.0	46.0	13.8	4.2
Blockton, Ala.....	55.5	44.5	11.4	3.6
Cartersville, Ill.....	55.3	44.7	11.2	3.5
Kemmerer, Wyo.....	45.3	54.7	10.7	3.2
Chimney Rock, Mont.....	55.6	44.4	13.0	3.0
Dawson, N. M.....	54.0	46.0	10.2	2.8
Eagle Pass, Tex.....	45.7	54.3	10.8	2.7
Caryneville, Wy. (sub-bituminous).....	58.6	41.4	6.8	2.5
East Creek, Wash.....	57.8	42.2	8.0	2.3

* This sample raised from the S. S. "Maine" after submersion 12 years in Havana Harbor.

² L. B. Eichengreen: Will Bituminous Coal Displace Anthracite and Coke as Generator Fuel? *Amer. Gas Assn. Monthly* (Jan., 1926) 8, 59.

³ H. C. Porter: Spontaneous Combustion. *Orig. Commun.* 8th Internatl. Cong. Appl. Chem. (1912) 10, 251.

Mr. Odell's paper brings out in general the trend toward use of a wider variety of coals in gas making. While water gas, through the cost reductions afforded by the bituminous generator and some concessions in B. t. u. standards, has taken on a "new lease of life," pending further rise in the price of gas oil, there are nevertheless unmistakable trends in many sections of the country toward coke-oven gas and other coal gas. New England operators and those of other sections where anthracite is at a disadvantage, believe in coke, and where coke can be sold for domestic and general fuel uses at a fair price, coal gas through the medium of by-product ovens and the modern retorts, will continue to gain favor at the expense of carburetted water gas, and will finally prevail as the favorite means of meeting at least the base loads of the gas industry.

The strong and growing sentiment throughout the country against smoke and the realization of its menace to health, is helping greatly to increase the use of gas and coke, the rational smokeless fuels for the future.

The effect of all this on the coal industry is likely to be a considerable widening of scope of the term "gas coals," the application of many Western and Midwestern coals for coke-oven use either wholly or in part and for bituminous gas generators, and the use of washed slacks in coke ovens. The diminishing supply of low-sulfur coals, and the necessity for low-sulfur content in gas for public distribution, will require increased use of washing and dry-cleaning methods, particularly on the slacks and small sizes, and a study of how best to combine sizing and cleaning methods to meet these new demands of industry.

W. H. BLAUVELT.—Are the shatter tests indicative of what would actually happen to the coal as it is ordinarily handled? Is that method of test representative?

H. C. PORTER.—For some uses I think it might be. The Canadian Bureau of Mines is starting some tests on a rumbler or a ball mill type of apparatus. I do not know which type of test would apply to most of the uses of coal.

W. H. BLAUVELT.—Since the coal is really broken by dropping in transit from the mine to the gas producer, perhaps the dropping test suggests itself as being indicative if the conditions are right.

H. C. PORTER.—It would seem so to me.

W. H. BLAUVELT.—I should think more so than the ball mill test.

H. C. PORTER.—It would seem that the drop would meet actual conditions more than the other.

J. GRIFFEN, Wilkes-Barre, Pa.—Are those cumulative sizes or individual?

H. C. PORTER.—They are individual, that is simply screening the 50 lb. of coal at the end of the four drops.

J. GRIFFEN.—What I mean is, are the individual sizes cumulative? Does the material through 2-in. include minus half?

H. C. PORTER.—The 2-in. includes both of the other sizes. However, the screening was done after the entire four drops.

W. H. FULWEILER.—Dr. Porter, have you had any experience on the effect of the size of the lump on these tests? We have encountered this question of drop tests. The coal that we would like to buy seems to be the best adapted to our purpose. It is a 3 by 6-in. lump. Some of the lumps are big and there is some evidence that the size of the original sample has an effect on the result obtained. Have you noted that in your work?

H. C. PORTER.—I have not made tests on any other sizes than 2-in. lump.

W. J. MURDOCK.—From practical experience, I can give you some idea of the amount of fines we reject on coal from Franklin County, Illinois, that we buy as 6 by 3-in. lump coal. When we handle this coal direct from a car to a track hopper, up a conveyor belt, over a shaking screen, into a bin, and then drop it into a larry car, from whence it is charged to the generator, we reject about 3 per cent. But if the coal is put in our stock pile by means of a track hopper, and a gantry crane and after remaining in stock for some months is reloaded into a car by the gantry crane and then handled in the same manner as fresh coal, the rejection is 11 to 12 per cent. However, with our process, we do not feel as much concern about the fines as formerly.

W. H. BLAUVELT.—If you had used 4 by 3-in. coal would you have had any less or more breakage as against 6 by 3-in.? I think that is Mr. Fulweiler's point. From one point of view the great, big lump comes down harder and hits the floor harder perhaps and might make more fines. On the other hand, it starts bigger and perhaps the amount of small coal at the end might be less on that account. I think experience is the only thing.

W. J. MURDOCK.—I have not had any actual figures or experience except watching the drop, and I think the breaking is along the natural lines of cracking of the coal. The majority of the fines chip off around the outer edge when you drop the coal, and handling it in a bulk, you do not get that except the chipping of the edge.

H. J. ROSE, Pittsburgh, Pa.—The point that Mr. Fulweiler brought up in connection with the shatter test on coal is equally vital in connection with the shatter test on coke, and yet it seems to be largely overlooked. The standard shatter test for coke calls for the use of coke, no piece of which will pass through a 2-in. sieve in any position. The coke may be full-length pieces, or it may consist of pieces which will just barely stay on a 2-in. sieve. The sensible and logical thing to do, is to make a complete sieve test before and after the shatter drops. Then you have something to go by. It is obviously unsatisfactory to start with coke of varying and unknown size and then carefully report to the last tenth of a per cent. the proportion of sizes found after breakage.

F. C. GREENE, Chicago, Ill.—We furnish a 3 by 6-in. lump coal for Mr. Murdock's operations. We would much rather furnish 4 by 3-in. coal. We do not make it, however. When we ship a car of 3 by 6-in. coal some thousands of miles or when we ship a car of 3 by 2-in., we will say, we always get a little more breakage in the 3 by 6 than we do in the 3 by 2-inch.

S. A. TAYLOR, Pittsburgh, Pa.—Before the mine run method of paying the miners was put into effect, I made some relative or comparative tests for results on the shipments of coal to the lakes, and shipping $\frac{3}{4}$ -in. screen coaling from the Pittsburgh district up to the lakes, meant dumping it into the cars at the mine, dumping it again into the boats at the lakeside, taking it out of the boats and dumping it on the dock and then rescreening it. I found that practically 33 per cent. of that coal went through a $\frac{3}{4}$ -in. screen at the docks, while the same coal from Logan County, West Virginia, went through the screens at the same docks one ton out of every eight.

F. HAAS, Fairmount, W. Va.—Our company (Consolidation Coal Co.) produces Elkhorn coal which has been used very extensively in this new process of utilization of coal. Every speaker so far has commented on the cost of the coal. If it is essential to this process that specifications as to size are closely adhered to it will result in a higher cost of coal. To take a size of 3 to 6-in. coal out of run of mine of gas coal is like taking the white meat out of a fowl. The remaining coal has a depreciated value and naturally the special size must have its proportional burden.

In certain classes of coal you will get but 15 per cent. of the 3 to 6-in. size, and there would be difficulty in getting the quantity desired from one particular mine.

W. J. MURDOCK.—We must experiment further with coal; lump coal is not just the thing to use. We are trying to find out how we can use slack coal in the generator. Apparently, the hindrance at the present time is the sacrificing of capacity in the water gas machine. The gas man wants to use any kind of coal so that when there is a price differentiation between a size coal and a run of mine, he will be able to take advantage of it in his economic operation. I have gone to a water gas machine and used 100 per cent. screenings. I have also used $\frac{1}{8}$ by $\frac{3}{4}$ -in. regular specified coal, and while our efficiencies over a 5-day period seemed to be practically the same as for a lump coal, we sacrificed considerable capacity.

W. H. FULWEILER.—I have an absolutely open mind on sizing. I am engaged in a survey of coals and if the operators have any ideas or suggestions as to how we can get a good coal cheaper I want to hear it. A gas works has to run—it has to make gas. We have to play safe. We do know we can get along with certain sizes of coal, and until we know more about this question we are probably going to pay more money and get less coal.

Whether it is due to the lower percentage of ash or to the fact that we can pick and choose with the slight differential that now exists between coke and coal, our average plants are showing a lower per pound consumption of fuel with bituminous coal than they did with coke. I will not say that this is altogether the case with bituminous coal, but it has been true during the past year; since we have improved some of our methods of operation, we have been using practically 3 lb. less coal per 1000 cu. ft. than we did with the coke.

W. H. BLAUVELT.—How much more oil?

W. H. FULWEILER.—The same amount of oil.

W. J. MURDOCK.—How much less capacity?

W. H. FULWEILER.—About 10 per cent. It is very difficult to get these figures down. Mr. Odell gave the impression that you would use more coke than coal. That may be true with some of the cheaper coals, but with the better grades of bituminous coal I feel that you can get your operations down and actually use less coal.

H. N. EAVENSON, Pittsburgh, Pa.—Mr. Odell did not mention the limit of sulfur. Is there not a limit on the sulfur?

W. J. MURDOCK.—It depends upon your purifying capacity to begin with, but I would say a coal that does not run over about 1.25 per cent. sulfur is usable in the ordinary plant. That is approximately what the Illinois coals run, from 1.1 to about 1.25 per cent. If the coal is good in every other way and there was ample purification capacity, I would not hesitate to use coal with a high-sulfur content. We have used Indiana coals with $2\frac{1}{2}$ or 3 per cent. sulfur, but we were fortunate enough to have good purification.

H. C. PORTER.—I suppose there would come into that problem the question of fixed sulfur and hydrogen sulfide. When the coal is too high in sulfur you cannot purify it with the purifying apparatus now used. There are methods being developed for taking out fixed sulfur.

W. H. FULWEILER.—The methods are on the market but there is no use in employing them because coal low enough in sulfur can be obtained. The sulfur that exists in carburetted water gas comes from two sources—the oil and the generator fuel. There is a very small amount of sulfur normally formed in coke. There is a little carbon oxysulfide and 4 or 5 grains of carbon bisulfide, and probably a few grains of hydrogen sulfide, but the oil is usually so low in sulfur that the sum total is way below the statu-

tory limits which were fixed to take care of coal gas. As you know, 30 grains are permissible. Now very rarely in water gas practice do you ever get over 12. There are some coals, however, that seem to form more hydrogen sulfide and more carbon bisulfide when used in the bituminous form, as we would naturally expect, than when used in coke, but our practice has generally been to use practically gas-coal types. Therefore, we have not been worried with sulfur.

We are now making some experiments on a small scale with the coal containing $3\frac{1}{2}$ per cent. of sulfur. When you make coal gas out of it you get a very high carbon bisulfide, also a very high hydrogen sulfide. I made a gas from a Cherokee coal that contained $6\frac{1}{2}$ per cent. sulfur. That was very poor fuel but it can be used.

D. B. REGER, Morgantown, W. Va.—It is difficult to regulate the kind of coal that comes out of the mouth of a mine. If there was a process by which could be used all of the coal that came out rather than to reject 75 per cent. of it, coal would be much cheaper. The cheapest coal that comes from the mine is of course the slack and the run of mine. Has anyone tried to briquette coal so as to have just the particular size of material that he wants in his furnace? Could not that be done cheaper than to buy the costly sizes of coal?

F. C. GREENE.—That is the meat of this whole question, for the coal man to tell the gas man how to use the coal. The trouble is that the coal man is not a gas man. However, there are a good many coal men who have got to make use of their finer sizes, and we look toward this gas-making with considerable interest. Our coal people are telling the gas man that he must devise his apparatus a little differently in order to get a cheaper generator fuel. The gas manufacturer replies that he is limited by what the State Commission allows him for returns, and that he has no appropriation for experimental research work. And the coal man feels that it is not his business, not at the present stage, anyway.

It is the solution of that portion of the problem that I am concerned in, and it would be desirable that the gas men and the coal men get together. It is manifestly impossible for a coal man to know about upblows and downblows and upsteaming and downsteaming and blowing into the atmosphere and steaming the atmosphere and into the holder. He does not want to be bothered with that. So he must collaborate with the gas man at some point, and until that collaboration is had, my view is that you are not only not going to get cheap coal for your generators, but you are not going to get it at all.

We cannot afford to sell you coal which costs us as much as we sell it to you for. If, on the contrary, you turn in and help us in the development of necessary apparatus the situation may be remedied.

W. H. BLAUVELT.—The coal operators may feel sure that the larger gas companies which have the facilities for experimental work are going to do everything in their power to so modify their operation and apparatus as may be necessary to use the cheapest types of coal. Among various industries, the gas industry is one of the few that has not increased its selling price materially since the days of low prices before the war. It has made wonderful advances. It has changed from a lighting to a heating industry, and many hundreds of thousands of dollars have been spent in experiment and research looking to that end. I am confident that the coal people may be quite sure that nothing will be left undone by the leaders in the gas industry to enable them to use, if possible, 100 per cent. slack coal. Then, I suppose, the producers will feel the price of slack coal ought to come up on account of the increased demand.

F. C. GREENE.—Absolutely.

W. H. BLAUVELT.—One wonders whether all the efforts of the gas man will in that event have gone for naught, because he is paying as much for slack as he did for run-of-mine or steam coal.

W. J. MURDOCK.—That is a study that is mapped out for the gas industry, a 3 to 5-year program of study of the economics of the gas industry, starting this year. We are not only studying it from the coal standpoint, but from the standpoint of heat quality, etc. Now let the coal men get right in with us and we will work hand in hand.

W. W. ODELL (author's reply to W. H. Fulweiler).—I recognize that there are existent conditions wherein the use of bituminous coal for the generation of water gas is used for making certain synthetic products; the limitation being based upon the methane content of the gas. One is not limited to present methods of operating nor to the use of present generating apparatus in gasifying coal; and with this knowledge I admit optimism in believing that, with a nominal differential in the prices of coal and coke, the former will be used. Obviously, efforts to use coal increase with an increase of this differential in favor of coal.

Mr. Fulweiler calls attention to the fact that the extent to which the blow-run can be used economically depends upon the relation of the price of coal to that of gas oil. This relation is positive, definite and very significant; it tells us that as the relative cost of gas oil increases with respect to that of coal, less blow-run can be used. There is then an optimum condition for each different cost relation and a point is ultimately reached where it is not economic to make a blow-run. As the use of the blow-run is limited—decreased—the gas-making capacity is also decreased and this factor enters into the economics of the process. These relations are significant because they indicate the necessity of developing other methods for using coal in the generator of a carburetted-water-gas set if its use is to continue as the cost of oil advances. In view of the often repeated intimation that gas oil may soon be higher in price and inferior in quality the latter statement should be full of meaning to the coal producers.

Fine-coal Cleaning by the Hydrotator Process

By W. L. REMICK,* NEW YORK, N. Y.

(New York Meeting, February, 1927)

THE hydrotator coal-cleaning process was developed as an economic necessity to meet the ever-increasing demand for an inexpensive method of cleaning coal down to the sizes ordinarily referred to as "dust." As the most urgent demand seemed to be in the anthracite field, all experimental work has been conducted in that territory during the past two years, with the exception of a few months during the strike of 1925-26, when some experiments were performed in the Alabama bituminous fields. The first commercial plant was constructed at the Middle Creek Colliery of the Philadelphia & Reading Coal & Iron Co., Pottsville, Pa. This plant is now operating.

APPLICATION

The field of application of the process is:

1. Cleaning anthracite fines, ordinarily described as "slush" or "silt." Slush contains all sizes from $\frac{3}{32}$ in. down to and including "dust." (Anthracite passing a $\frac{3}{32}$ -in. mesh is also referred to as No. 2 Barley and No. 4 Buckwheat.)
2. Cleaning sizes too fine for recovery by other methods.
3. Removal of coal from sand circulated in the Chance sand-flotation process.
4. Cleaning of bituminous sizes from $\frac{3}{8}$ in. down to "dust."
5. As equipment for the performance of the froth flotation process.
6. Other separating, extraction and washing processes which may be considered outside the realm of coal cleaning.

PRINCIPLE OF OPERATION

Fig. 1 shows the method of classification practiced under the Hydrotator patent.¹ In coal cleaning the machines are operated as "hindered settling" classifiers, the flow of solids and liquids being so controlled that the units may be combined in series.

* Manager, Hydrotator Co.

¹ U. S. Patent No. 1511643 (Oct. 14, 1924). Patents have been applied for on the various phases of coal cleaning by methods described in this paper.

In Fig. 1, pump *E* removes water and suspended solids from an upper level in the tank, and delivers this fluid mixture through arms *B* and nozzles *C* in uniform distribution over the tank bottom. These streams revert to a direct upward current in the tank. This upward current is supplemented by the solids and liquid entering the tank through the feed box *A*. It is obvious that the characteristics of the solids which reach the pump intake level depend upon:

1. Velocity of the upward current (hence the pump capacity), and volume of the feed.
2. Size and specific gravity of the solid particles.
3. Percentage of solids in suspension in the liquid.

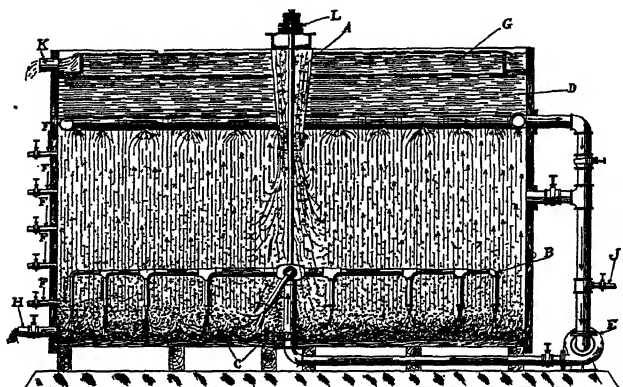


FIG. 1.—HYDROTATOR CLASSIFIER.

THE COUNTER-CURRENT COAL-CLEANING PROCESS

The process must necessarily be performed by two or more hydrotator classifiers in series. The number of classifying units depends upon the scope of the washing problem as determined through a study of the properties of the coal to be treated. The factors which determine the design are, for any given coal:

1. Range of sizes to be handled.
2. Ash reduction desired and percentage ash in original coal.
3. Tonnage to be handled.

The first two factors are interdependent. It is found practicable in the treatment of anthracite from the lower Schuylkill region, to reduce the ash content from 30 to 10 or 12 per cent. while treating material ranging in size from $\frac{3}{32}$ in. down to 65-mesh. If a similar reduction over a greater range of sizes is desired (for example, from $\frac{3}{16}$ in. down to 65-mesh), it becomes necessary to introduce a $\frac{3}{32}$ -in. screen after the

first stage of cleaning. This removes the $\frac{3}{16}$ to $\frac{3}{32}$ -in. coal, allowing the sizes finer than $\frac{3}{32}$ -in. to pass to the next stage of cleaning.

In Fig. 2, the first tank (2), into which the raw coal is fed, may be considered a "roughing cell." Using again the $\frac{3}{32}$ -in. to 65-mesh anthracite as an example, assume it is theoretically possible to make a 60-per cent. recovery of coal. The feed is split two ways in the first or "feed cell." Half of the material by weight, consisting of about 80 per cent. of the recovered coal, overflows to the second cell (3). The other half, containing the refuse and 20 per cent. of the coal, is discharged from the bottom of the first cell to the sump (1-S), from which it is pumped into the refuse cleaning cell (1). The coal overflows, joining the original feed, whereas the refuse is discharged from the bottom of the refuse cleaning cell.

From the second and third cells (3 and 4) middling products are returned to the first and second cells respectively through their respective sumps and circulating pumps (2-S and 3-S). The final clean coal overflows to a "dewatering" screen (5) which removes the water and solids finer than 65-mesh, or whatever the predetermined lower limit of sizes may be.

In Fig. 2 the feed cell (2) may be regarded as a "roughing" cell and the other two (3 and 4) as "finishing" cells, the number of finishing cells depending upon the desired reduction in ash content. It is found that, in reducing the ash content of slush from 30 to 12 per cent., about 70 per cent. of the total reduction takes place in the first stage, about 10 per cent. in each of the next two stages and the final 10 per cent. is effected by the removal of the high-ash, -65-mesh material on the "dewatering" screen.

Agitation

To maintain the desired upward current in the cells, a side overflow (6) is provided on each cell, except the small refuse cleaning cell. This overflow goes to the sump under the same cell, joining there the "underflow" from the next cell following the series; for example, overflow from (3) and underflow from (4) go to sump 3-S. Control of this overflow provides the regulation necessary to maintain complete suspension of solids in the cells.

Water Supply

It is necessary that the feed shall contain no more water than is necessary for the proper control of the process. The feed water is supplemented by the addition of water to the refuse cleaning cell (1). This provides the proper dilution for the "underflow" from the feed cell as well as control of the settling conditions in the refuse-cleaning cell itself. This make-up water is added to the sump (1-S) under the refuse cell.

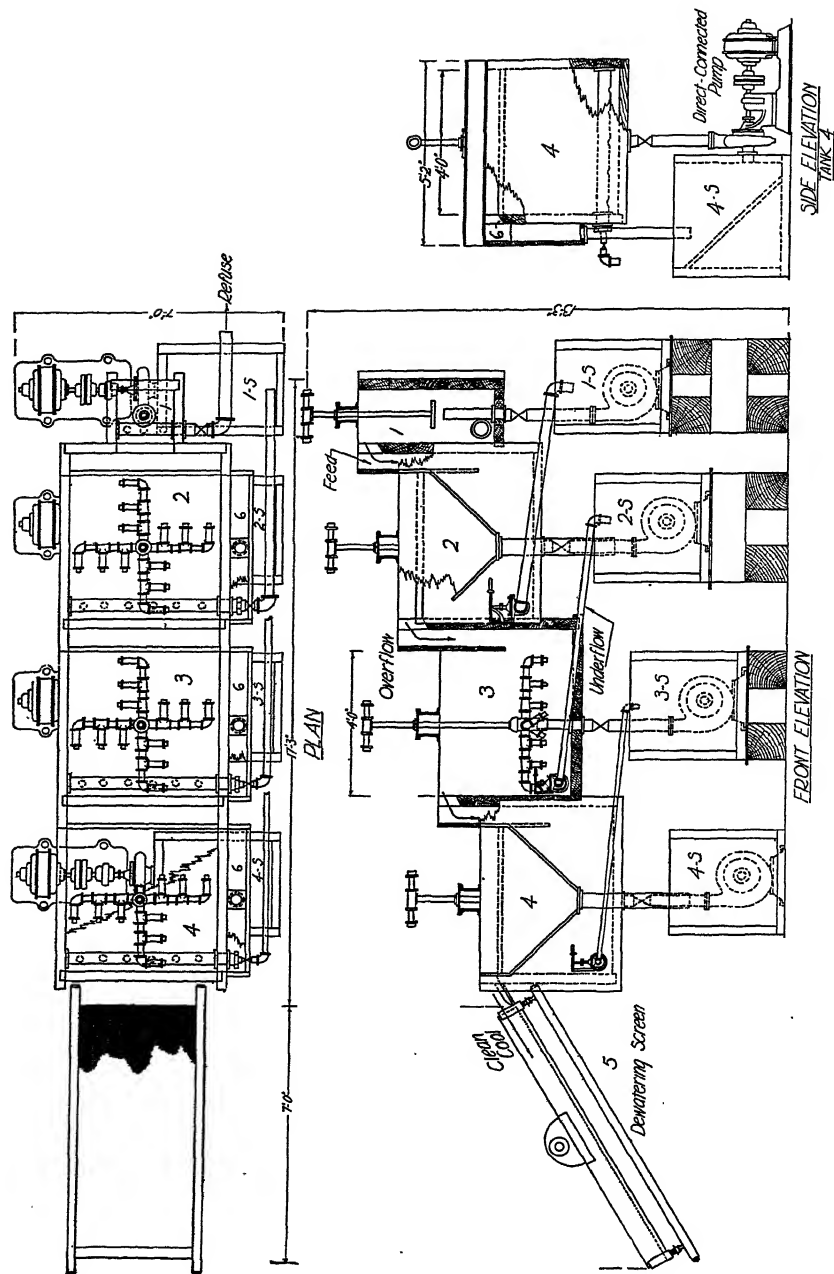


FIG. 2.—COUNTER-CURRENT COAL-CLEANING PROCESS.

FROTH FLOTATION OF COAL

Although a large amount of research work has been done both by companies and individuals in connection with the cleaning of coal by what is usually known as "froth flotation," there is very little published information available. It would seem advisable, therefore, to explain briefly the principles involved in the adaptation of froth flotation to the cleaning of coal. Richards and Locke² give the following definition of flotation:

Flotation is based on the phenomenon that some minerals are easily wetted by water while others are not so easily wetted, or, to express it in another way, gas bubbles attach themselves more readily to some mineral particles and, when attached, resist being displaced by water. The result is that under suitable conditions the mineral particles which are not easily wetted either are buoyed up as a film on the surface of the liquid by a thin layer of gas between the particles and the liquid or are supported in a layer of froth above the liquid, this froth consisting of a mass of fine particles and gas bubbles. In general, sulphide minerals, native metals, sulphur, and carbonaceous or graphitic materials float; gangue minerals, such as oxides and silicates, sink. Thus we have a separation in which specific gravity plays no part and in which usually the floating mineral is heavy and the sinking mineral is light.

Coal comes under the head of "carbonaceous——materials."

The effect of the affinity of oil for coal is accentuated in actual operation by the fact that coal is much lighter than slate, whereas in ore flotation the floated minerals are much heavier than the non-floating gangue minerals. This accounts for the fact that larger sizes may be floated in coal than in sulfide flotation.

Another important difference between coal and sulfide flotation is the difference in the comparative bulk of the floated minerals. Whereas the metal content of sulfide flotation feed seldom runs higher than 7 per cent. and, in the case of copper, is usually less than 2 per cent., we have in the flotation of coal the reverse condition. The refuse or heavier constituents comprise usually less than 25 per cent. of the feed, whereas the rest, or coal, has to be handled in the froth.

It is obvious that the designer of a machine for flotation of coal must take account of two important factors:

1. Floated material varies in specific gravity from 1.3 to 1.7, whereas the tailings, or refuse, range from 1.7 to 2.6 specific gravity.
2. Ordinarily the bulk of floated material is at least three times that of the refuse. This is in direct contrast to conditions found in sulfide flotation plants.

ADAPTATION OF HYDROTATOR TO FROTH FLOTATION

The flow sheet shown in Fig. 2 is changed slightly for adaptation to froth flotation. The coal is fed into the last cell in the series (4). Under these circumstances the other cells represent successive stages of reclean-

² Richard and Locke: Textbook of Ore Dressing. 2nd ed., 264.

ing applied to the refuse from the feed cell. Provision is made for the removal of the froth from the cells and for the introduction of flotation agents and air through the pumps. The refuse is discharged as in the counter-current system. The number of cells required is dependent upon the properties of the coal.

It is conceivable that it might become necessary or advisable to grind between stages of flotation. This would be done under ideal conditions, as the underflow of each cell is classified very efficiently. In case such regrinding is practiced the underflow from cell (4) is ground before discharging to sump (3-S). This releases coal from combined particles of coal and slate. The coal is floated in cell (3) and a classified product is discharged to sump 2-S after passing a grinding stage. This is in accordance with the approved practice in coal washing and ore dressing, which makes it advisable to recover valuable material as soon as its physical condition makes recovery possible. Recovery between stages of grinding has long been an established practice in ore dressing.

Control of Conditions

Most of the excess water is, in this instance, discharged with the refuse, only enough overflowing the cells to carry the froth. Incidentally, this overflow is the controlling factor in determining the cleanness of the floated coal. If the overflow is increased, a combination of the counter-current and the purely flotation processes is attained. This controls not alone the cleanness of the product but the size of coal cleaned.

The cleaning effect in froth flotation practice is to a certain degree independent of the range of sizes treated. The result is that the "dust" sizes are recovered as clean coal. Here the screen is more nearly a dewatering screen in fact, even though impurities entrained in the froth are eliminated through the screen. The underflow from the screen is returned to the recleaning-cell sumps, to conserve coal and oil.

Flotation Agents

The crudest kind of tar oil may be used in quantities found to be less than 0.1 per cent. by weight of clean coal produced.

The adaptation of the hydrotator principle to froth flotation is susceptible to many variations and constitutes a subject so broad that it is considered proper to treat it only in outline in this paper.

CLEANING BITUMINOUS COALS

It is particularly true of bituminous coals that varying percentages of the total ash content will be found in the different sizes. An outstanding example is found in Alabama. The Black Creek seam has a fire-clay bottom, and inevitably fire clay gets into the mined coal. This

tendency will increase as machine mining practice is increased, and mechanical loading becomes more popular. The ash content increases as the size decreases, the finest or "dust" containing the highest percentage of ash. Nearby, in the Pratt seam, conditions are reversed. The impurities are less friable than the coal with the result that the smallest sizes are cleanest.

As far as the hydrotator coal cleaning process is concerned, this variation influences only the dewatering stage. If the fines are high in ash, a screen is used; if they are low in ash, a thickener is used, followed by whatever final dewatering process may be found suitable. The dewatering method depends largely upon the subsequent treatment of the coal.

Handling Clean Coal and Refuse

When a dewatering screen is used, the coal is dry enough to be handled on a belt conveyor. The refuse may be handled by an elevator, scraper conveyor or pump, according to the treatment following and the ultimate destination of the material.

Power Requirements

For a unit producing from 100 to 200 tons per day of clean coal the power required, aside from that used in handling materials to and from the plant, is from 8.5 to 13 hp. for the operation of circulating pumps and dewatering equipment.

Labor

One man per shift can operate from 1 to 5 units. This does not include preparation and handling feed and disposal of products, but merely the operation of the cleaning plant proper.

Plant

A 100 to 200-ton unit occupies a space approximately 24 by 7 by 13 ft. high. If it is operated as an accessory to the treatment of larger sizes, as in anthracite practice, it is usually possible to find room in existing washery buildings for a hydrotator plant large enough to handle all of the fines produced.

If operated as an independent unit, as in the treatment of coking coals after crushing to $\frac{1}{4}$ or $\frac{3}{8}$ -in. size, it is necessary to provide a single-story building with the necessary bin capacity on both feed and discharge ends. A side hill location with crushing machinery above the feed bins is desirable. In any case the buildings are of the most inexpensive type, as no part of the equipment is suspended in such a manner as to require structural supports. Everything rests upon simple concrete foundations.

OPERATING RESULTS

The Counter-current Process

Table 1 is a typical screen analysis of "raw" anthracite "slush," or "silt." This particular sample was taken in the Southern Schuylkill region where the percentage of fines produced is greater than in the northern fields.

TABLE 1.—*Screen Analysis of Raw Anthracite Coal*

Mesh	Weight, Per Cent.	Cumulative Weight, Per Cent.	Ash, Per Cent.
+ 20	15.3	15.3	19.9
+ 48	35.2	50.5	26.3
+ 65	9.1	59.6	33.2
+100	7.4	67.0	35.4
- 100	33.0	100.0	44.8
100.0		Calculated Ash = 32.7	

It will be observed that 33 per cent. of the solids in raw slush are finer than 100 mesh, though the percentage of pure coal is much lower than in the coarser sizes.

The usual procedure in anthracite preparation is first to thicken the slush, which averages from 5 to 10 per cent. solids by weight. Thickeners are of various types, but are almost invariably operated as classifiers, so designed as to overflow as much of the fine material as may be consistent with the disposal of water and the subsequent treatment of the fine coal. The result is the retention in the slush recovered of about 5 per cent. of "finer than 100 mesh" material.

This elimination represents a loss of coal that is unavoidable unless froth flotation is employed in its recovery. It is not generally regarded as economically feasible under present market conditions, to recover these "slimes." The difficulties encountered in dewatering these extremely fine sizes, and the resultant transportation problems involved are, at least in the anthracite field, almost insuperable. In the case of bituminous coking coals, where fine crushing before cleaning is becoming common practice, transportation is usually accomplished by calking the cars and by loading the fines mixed with the coarser sizes. It is obvious that if anthracite fines are to be loaded in cars they may not be mixed with the coarser sizes. The $\frac{3}{32}$ -in. anthracite is now being transported in ordinary "gondola" cars.

The steady increase in ash content as sizes decrease may be observed in Table 1. This is also noticeable in the clean coal obtained by the hydrotator process, as shown in Table 2.

TABLE 2.—*Coal Cleaned by Counter-current (Hydrotator) Process*

Mesh	Weight, Per Cent.	Cumulative Weight, Per Cent.	Ash, Per Cent.
+20	12.4	12.4	7.37
+28	40.6	53.0	9.76
+48	20.8	73.8	12.72
+65	25.2	99.0	20.33
—65	1.0	100.0	28.66
100.0		Calculated Ash = 13.02	

Only 1 per cent. of material finer than 65-mesh remains in the clean coal. This simplifies the dewatering problem, as such a product will drain in the cars to 8 or 10 per cent. moisture. It may be taken as a rough rule that each per cent. of — 100-mesh material will add 1 per cent. to the moisture content of the coal when shipped.

Thirteen and two-one hundredths per cent. ash is somewhat lower than is found in the anthracite "steam" sizes now being shipped from the region from which this sample was taken. About 10 per cent. ash is the cleanest coal now being shipped from any part of the anthracite region. These figures are given to avoid odious comparisons with results obtained in washing bituminous coals.

RESULTS BY FROTH FLOTATION

Previous to the development of the hydrotator coal-cleaning process, extensive experiments were performed, using the same type of hydrotator as a flotation machine. Tests were made on both anthracite and bituminous coals. Treating the same coal shown in Table 1, the results in Table 3 were obtained:

TABLE 3.—*Screen Analysis of Floated Anthracite Coal*

Mesh	Weight, Per Cent.	Cumulative Weight, Per Cent.	Ash, Per Cent.
+ 20	12.5	12.5	6.9
+ 48	45.0	57.5	8.4
+ 65	11.3	68.8	8.9
+100	12.5	81.3	9.1
—100	18.7	100.0	11.3
100.0		Calculated Ash = 8.7	

There is a decided contrast between this clean coal and that shown in Table 2, which shows only 1 per cent. finer than 65-mesh, whereas the floated coal has 31.2 per cent. finer than 65-mesh and 18.7 per cent. finer

than 100-mesh. The ash content is 8.7 per cent. as compared with 13.02 per cent., in Table 2. Although it is generally considered that the "inherent ash" in this coal is 8 per cent., the coarsest size in this sample contained only 6.9 per cent. ash.

This sample was dewatered on one of the well known continuous vacuum filters. The filtered coal contained 19 per cent. moisture. All of the sample was finer than $\frac{3}{32}$ in., which is approximately the equivalent of a 10-mesh Tyler standard screen.

FLotation OF BITUMINOUS COAL

Table 4 represents an attempt at a comparison of the results obtained in jiggling bituminous coal and floating the same coal. This coal has an assumed inherent ash of about 2.25 per cent.

TABLE 4.—*Comparative Screen Analysis of Raw, Jigged and Floated Coal (Alabama Bituminous)*

Mesh	Weight, Per Cent.	Ash, Raw Coal, Per Cent.	Ash, Jigged Coal, Per Cent.	Ash, Floated Coal, Per Cent.
+ 20	71.5	19.65	3.87	1.60
+ 48	19.6	24.80	11.20	2.79
+100	4.8	28.05	15.97	3.58
+200	2.5	29.65	19.20	5.20
-200	1.6	34.89	23.35	8.64
Totals.....	100.0			
Ash Averages, Per Cent...		21.58	6.58	2.20

Note: Using the highest results obtained by flotation, the result would be 2.90 per cent. ash, instead of 2.20, as shown. It is not claimed that in actual practice a floated coal could be made consistently with an ash content as low as that shown. On this particular coal it should be possible to keep the ash below 3 per cent. in commercial practice.

With a coal crushed to pass a 1-in. round hole screen, from 45 to 55 per cent. will pass the $\frac{3}{32}$ -in. round hole screen.

Tonnage

The hydrotator coal-cleaning process will, in a single unit, using 4-ft. cells, produce from 100 to 250 tons of clean coal per 8-hr. shift, depending upon the ash content of the raw coal and the desired reduction in ash content. The recovery is close to the theoretical maximum, but exact figures are not yet available. Flotation will give somewhat higher tonnage.

Costs

Exact figures cannot now be published, but a liberal estimate indicates a total cost of production of less than 10 c. per long ton of clean coal.

CONCLUSION

An attempt has been made in this paper to stress the need of an inexpensive method of cleaning low-priced coal. The question of plant cost and the resultant capital cost as applied to the cost of producing clean coal is important in any process involving the treatment of coal. There is so little margin in many cases between cost and selling price, that a matter of 10 c. difference in cost of cleaning may decide in favor of operation of a mine that might otherwise lie idle for years.

This item of plant cost becomes paramount when the coal to be treated has in the past been considered unmarketable, as in the case of anthracite slush. Unless briquetted, this fine anthracite must compete with bituminous coal, which itself cannot stand a high-treatment charge.

It now seems apparent that, by the treatment of the finer sizes of bituminous, it is becoming possible to place on the coking-coal market many coals that have hitherto been considered uncleanable to the required degree.

ACKNOWLEDGMENTS

Thanks are due to H. E. Muelhof, master carpenter of the Philadelphia & Reading Coal & Iron Co., for invaluable aid in the development of the hydrotator coal-cleaning process. A. G. Overton, general superintendent of the Alabama By-products Co., and H. B. Stockett, manager of the Chas. M. Dodson Co., rendered very great assistance in the loan of their facilities for the adaptation of the hydrotator process to the cleaning of anthracite and bituminous coals by froth flotation.

DISCUSSION

H. C. PORTER, Philadelphia, Pa.—How much refuse is discarded?

W. L. REMICK.—Sixty per cent. is a good recovery in the anthracite field.

H. C. PORTER.—Is that on the froth flotation?

W. L. REMICK.—No, recovery is a little better than that on the froth flotation.

W. H. BLAUVELT, New York, N.Y.—Do you mean 60 per cent. of the carbon in the coal?

W. L. REMICK.—No, 60 per cent. by weight.

W. H. BLAUVELT.—How much carbon in the ash?

W. L. REMICK.—That depends upon how clean a coal is desired. The ash in the refuse would probably run above 50 per cent. By oil flotation there would be no difficulty in getting it up to perhaps 60 or 65 per cent. We cannot give definite figures on that.

E. T. CONNER, Scranton, Pa.—Have you used this process in any other sizes than the slush, so-called?

W. L. REMICK.—No, not yet on anthracite. That is, experimentally we have cleaned bituminous coal up to $\frac{3}{8}$ inch.

E. T. CONNER.—But in anthracite you have confined your experimental work to the so-called slush?

W. L. REMICK.—We have cleaned larger size in experimental work but not in actual practice.

E. T. CONNER.—For the enlightenment of those who do not know anthracite, will you specify the size you mean when you say barley?

W. L. REMICK.—It seems to me that it is $\frac{3}{16}$ in. There is so much difference of opinion on that point that I am confused myself.

E. T. CONNER.—As you have not applied your apparatus to anything but waste material it cannot, as I see it, be compared with devices or processes now in use in the anthracite region for the preparation of domestic sizes.

W. L. REMICK.—No, it cannot. That question of waste material though is a matter of opinion.

E. T. CONNER.—The so-called barley and smaller sizes are, generally speaking, of relatively small value per ton. This then would be a reclamation process.

W. L. REMICK.—We have made absolutely no pretense of handling the large sizes.

A. W. HESSE, Nemaquin, Pa.—Is the Alabama plant still in operation, that is, has it been applied practically?

W. L. REMICK.—That was merely an experimental run. We had a machine built for experimental purposes and it is now located in the Mining Laboratory of Carnegie Institute of Technology. It is, however, a 3-ft. tank and in actual practice we only use 4 and 5-ft. tanks, so we really do not get "laboratory results."

A. W. HESSE.—Is that plant now being used for experimental purposes?

W. L. REMICK.—Yes.

W. H. FULWEILER, Philadelphia, Pa.—What do you mean by "tar oil"? So many oils are made from tar. Is it a straight distillate? We were in the tar distilling business at one time and still are. We made 128 different products out of tar that can all be called tar oil. I would like to know which one had worked.

W. L. REMICK.—The tar oil I referred to was the crude oil and the only product called tar oil by the Alabama By-products Co. at Birmingham. The only use they had for it was in competition with fuel oil. It cost the company 7 c. a gallon.

H. J. ROSE, Pittsburgh, Pa.—For flotation generally, the light or medium creosote oils are desirable, and, so far as I know, they are also desirable in coal flotation. Creosote oils from coal tar are very satisfactory as frothing agents. The light oils from water gas tar are not good frothing agents while Scotch blast-furnace oils are intermediate in character and not as good as coal-tar creosote, which I understand is best.

W. H. FULWEILER.—There are about eight different specifications. That may add to the complications.

L. W. HUBER, Pittsburgh, Pa.—We went to Mellon Institute for a sample of oil that could be used, and they gave us No. 2 distillate, creosote tar oil.

W. H. FULWEILER.—Does it grade 2?

L. W. HUBER.—No. 2 is what they called it, whether that is the grade or the specific gravity I do not know. I think almost any tar oil will do that is not too heavy.

A. ALLEN, JR., Chicago, Ill.—I think the treatment of very fine sizes of coal has got to come, especially in preparation of bituminous coals for coking purposes where the loss of fine coal is just as serious a matter as the loss of large coal. Every process must be judged by the possible commercial recovery of coal at a certain standard ash or sulfur content. Mr. Remick, can you give us any information regarding the ash content of the refuse in the different sizes?

W. L. REMICK.—You will invariably find that the sizes vary in the same way that they do with the feed and in clean coal. The ash content is highest in the finest sizes.

A. ALLEN.—This being a hindered settling proposition there is a different separation made on every size, and the efficiency of this separation at any particular size would be measured by the ash content in the overflow and in the tailings. Can you give me any idea as to the ratio of size which can be efficiently handled in a single apparatus.

W. L. REMICK.—I believe the sizes that I mentioned, take anthracite as an example from $\frac{3}{32}$ in. down to about 60-mesh, represent just about the maximum range that you can expect to get in one stage.

A. ALLEN.—That would give a certain size ratio which you can handle efficiently.

W. L. REMICK.—Whether that would apply as you went up the scale I am not able to say at present.

A. ALLEN.—Could you treat sludge effectively from 48-mesh down to 200-mesh, which would be a ratio of 4 to 1?

W. L. REMICK.—I believe that it could be done, although we have not tried to do it on a practical scale. I have just one clue that might lead to that. We attempted to dewater in a hydraulic classifier and fell down because we found that we were getting the cleanest coal in the overflow of the classifier. On this particular job on anthracite, we got coal running about 12 per cent. ash, which was finer than 100-mesh.

A. ALLEN.—In one case that I am familiar with, we are getting an overflow from a Dorr tank with 18 per cent. ash in the solids and an underflow with possibly 25 per cent.

W. L. REMICK.—That is to be expected.

A. ALLEN.—In some cases it happens that way, although the general rule is the other way about.

W. L. REMICK.—I do not believe that the process could be applied on a practical scale for handling those sizes.

A. ALLEN.—I will pass to another question about froth flotation. I have been very much interested on account of the great efficiency of this process. Some tests I have seen in a coal having 22 or 23 per cent. ash show a froth having 10 per cent. ash with tailings containing 60 per cent. ash. No gravity separation that I am familiar with can give anything like those figures on fine coal. When oil is used we are able to get a very much higher recovery with a purer coal.

W. L. REMICK.—That is due to the principle of operation.

A. ALLEN.—That is what interests me particularly. How is it possible to handle the froth commercially? Do you screen it or do you take it just as is? It will contain about 50 per cent. water as it comes off. Can you screen it, before the froth breaks down?

W. L. REMICK.—I can give you an actual case as far as anthracite is concerned. The first screen analysis I gave you is raw anthracite slush from $\frac{3}{32}$ in. down, containing about 33 per cent. —100-mesh material. We got a floated coal using that same feed containing about 18 per cent. — 100. That was sent to the United Filters Co. They dewatered it and got the moisture down to 19 per cent.

A. ALLEN.—Before the froth breaks down it can be handled readily on the conveyor, I presume, without attempting to dewater it?

W. L. REMICK.—Yes. We actually got 10-mesh, that is, about $\frac{3}{32}$ -in. sizes in our froth. Whether or not that was by true oil flotation is open to debate.

A. ALLEN.—Did you have a rising current in the apparatus?

W. L. REMICK.—No, without the rising current. That is, the test was done in a machine similar to the one I first described, the one which has the quiet zone above. There is a rising current in the lower part which is contrary to practice in the machine that is ordinarily used in sulfide flotation. There usually is a down current, and the sulfide is floated away from it. Nineteen per cent. was the best they could do by filtering. You can heap that coal up on the ground as it comes off the machine. Simply pile it up and let it stand for 24 hr. and it still has 25 or 27 per cent. moisture.

A. ALLEN.—Do I understand that the mixture in the froth comes down from 50 to about 25 per cent. by natural drainage?

W. L. REMICK.—Yes. It is very difficult to get it by drainage below that. However, each consumer has to decide for himself whether it is worth while to take coal with that percentage of moisture. There are filters developed that can do better than that. I believe the Loughlin filter has got the same material down to about 14 per cent. It depends upon how much fines there is in it.

A. ALLEN.—That depends on fines, and the general result depends on the proportion of your fine material as you mix it back with larger sizes of coal. You may have a water content of 15 or 20 per cent. in your material under 60 mesh, and still not have an objectionable water content in the aggregate.

W. L. REMICK.—There is an interesting point as to the breaking down of the froth. The froth was raked off from the froth flotation machine, into a box, and half an hour later 90 per cent., probably more, had settled at the bottom and was entirely free from froth, and by comparatively gentle stirring you could break up most of the balance to a great extent. The extremely fine stuff, however, is rather persistent.

A. ALLEN.—Have you any flotation plants with oil in operation?

W. L. REMICK.—We have not.

A. ALLEN.—Only the experimental unit at the Carnegie Institute?

W. L. REMICK.—Only the one unit.

A. ALLEN.—You can run that either with oil or as a counter-current proposition?

W. L. REMICK.—Yes.

Outbursts of Gas and Coal at Cassidy Colliery, Vancouver Island, British Columbia

By R. R. WILSON,* AND ROBERT HENDERSON,† CASSIDY, B. C.

(New York Meeting, February, 1927)

THE Cassidy Colliery operated by the Granby Consolidated Mining Smelting & Power Co., Ltd., is situated about 9 miles in a southerly direction from the city of Nanaimo on Vancouver Island. The coal seam worked, which is called the Douglas bed, outcrops in the western part of the property, dipping easterly, and is opened by a slope 7 by 14 ft. in the clear, and is timbered by 12 to 14-in. framed sets of timber, 4 ft. center to center. A separate manway provides for a traveling road and at the same time forms an intake airway.

The general average of the dip of the seam at Cassidy is 18° , and varies in thickness from 1 to 25 ft., averaging about 8 ft. The main slopes follow straight down the dip a distance of 4845 ft. At the bottom of the main slope two rock tunnels were driven a distance of 1500 ft. to the south, cutting through a fault, and opening up an area in the same seam which is known as No. 7 South Section. The coal is found in this section at a vertical depth of 1560 ft., and is very soft and friable.

The immediate roof is a strong, sandy shale overlain by massive sandstone while the floor is a light-colored shale, hard when freshly exposed. It does not soften materially on exposure.

In the folding of the coal measures a great thrust caused a movement along the coal seam as indicated by the slickensided nature of the coal. The floor has buckled and this has rolled and squeezed the coal so that with rare exceptions it shows no normal bedding but curved slip planes, which frequently start at right angles to the floor, curving upward and then in the direction of the dip. As such planes are slickensided it is probable that there is a coating of clay which is more or less impervious to gas.

OUTBURSTS OF GAS AND COAL

Outbursts of gas and coal have occurred in this mine previous to 1921, but not until June 14 of that year was a complete record kept of them. No outbursts occurred above the fourth levels, or, in other words, until a vertical depth of about 900 ft. was reached. Since that time quite a

* Consulting mining engineer.

† Mine manager.

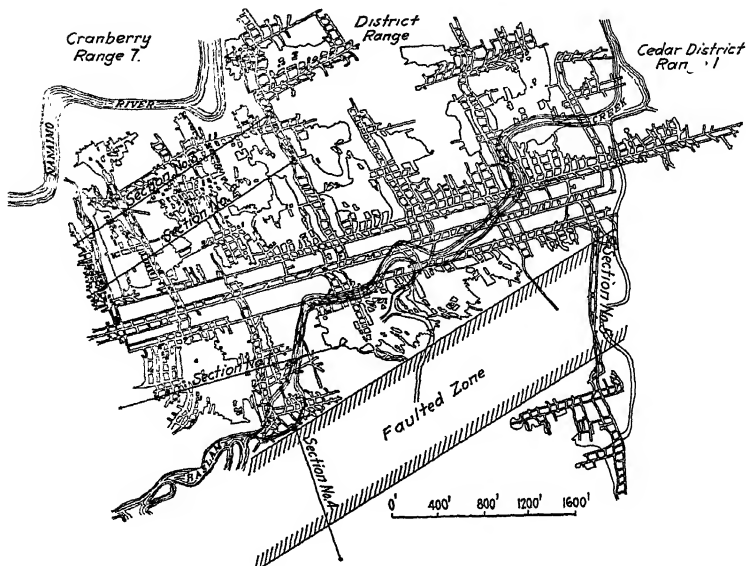


FIG. 1—PROGRESS MAP OF GRANBY COLLIERY No. 1, SHOWING MONTHLY DEVELOPMENT.

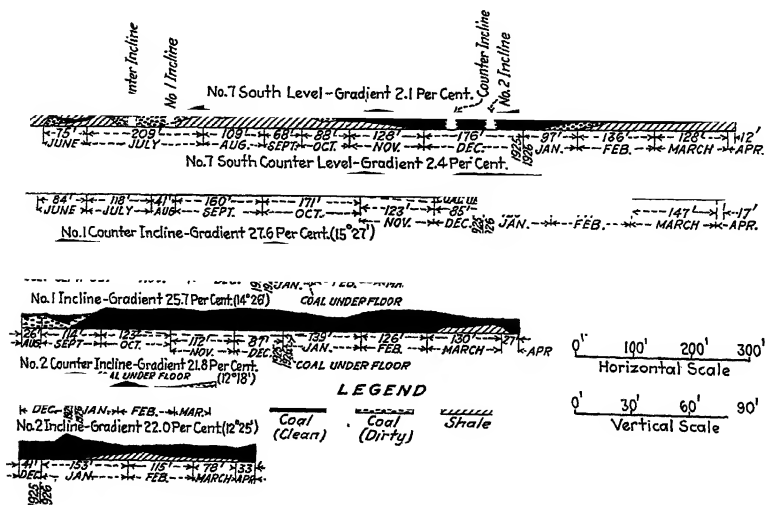


FIG. 2.—SECTIONS OF DOUGLAS SEAM IN No. 7, SOUTH DISTRICT.

number of outbursts have occurred with more or less violence. The total number of such outbursts recorded is 268.

The blow-out areas are restricted in size. The pressure is enormous, but the gas would appear not to be instantly liberated. The action of many of the outbursts has been watched and described as pushing the coal before it in rolls like the waves of the sea.

On April 7, 1926, an outburst of coal and gas occurred in the main level of No. 7 South Section, resulting in the death of two miners who were working in the place. One month previous to this accident it was agreed between the management and the Mines Department to increase the diameter of the advance holes from 2 to 6 in. At the time of the accident four holes 6 in. in dia. and 12 ft. long were drilled in the place. The seam was 3 ft. thick and 3 ft. of bottom rock was taken up to make the required height for haulage.

The Government Mine Inspector and the overman visited the place on the morning of the accident. At that time the two miners were mining the coal and the chief engineer and his assistant were in the same level extending the survey. The four drill holes were examined carefully by the Inspector who found that only one hole gave off just sufficient gas to show about $\frac{1}{2}$ in. cap on the flame of his safety lamp. He could not detect any gas coming from the other three holes. The place was 16 ft. wide and well timbered, and there were no signs of any roof, floor or side movement; yet in less than 10 min. after the Inspector and the overman had left, an outburst occurred with the fatal results already mentioned. The two engineers were eyewitnesses of the accident, the assistant engineer being only 12 ft. from the face and the chief engineer 60 ft. back. The former had his back to the working face when the blow-out took place. He says "no warning was given." The chief engineer was looking through his instrument 60 ft. back and saw what he described as a flow of fine coal coming in rolls or waves. In this particular case very little gas was given off as the Inspector and the overman were able to reach the face less than 5 min. after the blow-out happened.

THEORIES ADVANCED FOR OUTBURSTS

It is a matter of common knowledge to mining men that blow-outs or outbursts of gas are likely to occur in the neighborhood of faults, and it is sometimes supposed that faults act as reservoirs for gas, and that it is from these reservoirs that the blowers are fed.

At Cassidy there do not seem to be any such reservoirs or blowers, because an order issued by the British Columbia Mines Department compelling at least three drill holes to be bored in advance of all working faces below the sixth levels was strictly carried out, and although some miles of such drilling had been done not once did any of the holes strike a pocket of gas or indicate any gas pressure. The following theories have

been advanced as to probable causes: (1) Gas pockets in the seam proved that no cavities existed. (2) Gas existed in cavities in the coal seam in liquid form, proving that no cavities existed and no pockets of gas were released by drill holes. (3) The release of the pressure under which the gas is held in the pores of the coal accounts for the sudden outbursts of coal and gas. The latter theory is advanced and is accepted locally as the most probable. It is confirmed to a certain extent by the fact that most of the outbursts have been brought on by the miner picking or mining the coal, or by shot-firing.

It has been proved at the Cassidy Colliery: (1) That boreholes 2 in. in dia. and 20 ft. long and boreholes 6 in. in dia. and 12 ft. long failed to release any gas pressure, or indicate any cavities in advance of the working faces. (2) That at no time did any such boreholes when examined indicate gas pressure although frequently a cap of gas would be shown on the flame of a Wolf safety lamp. (3) That coal faces exposed in the air current for a period exceeding 2 years were as liable to an outburst of gas and coal, provided operations were commenced on it, as freshly-opened work in the virgin coal. A chain pillar on the main slope was being split for haulage purposes 2 years after opening it and a few small pushes occurred.

Where these outbursts occur they are preceded by a humming noise like the workings of a jackhammer drill, or the firing of a machine gun, and are accompanied by a rumbling noise similar to a runaway trip of cars on an incline. A great quantity of fine coal is blown or pushed out covering the roadway many feet in thickness on the floor until the pressure exhausts itself. The blow-outs do not come from the same direction, but may come at any angle from the working face; in fact, in one instance, the side of a pillar blew out after the face had advanced beyond the point 50 feet.

The coal that is dislodged is very fine and scarcely a piece the size of a nut can be found in it, but it is usually moist enough to mold in the hand.

PRESSURE EXERTED

The great pressure exerted during these outbursts can be gaged by the fact that a full car of coal ($2\frac{1}{2}$ tons gross weight) was pushed for a distance of 20 ft. up a gradient of 18° . On that occasion six sets of timber also were pushed out a distance of 24 ft. from the working face where the blow-out occurred, and a number of posts between the roof and floor that could not be pushed out were split and broken. Only in a few instances where the coal has been pushed out as described has the roof caved in. Usually the roof stays up until the coal is all loaded out and new timbers are set. The effect is more like that produced by making a heavy blast with black powder. The appearance of the loose coal and dust as it is dug away from the blow-outs is a honeycomb structure.

It is all fine and loose, and between the top of the loose coal and the roof is a space varying from 6 in. to 1 ft. It is sometimes possible to see with the aid of an Edison electric miner's head lamp a distance of approximately 40 ft. along the top of the loose coal. This space is evidently formed by settlement of the coal.

REMEDIES SUGGESTED

Various mining men have suggested as remedies for outbursts: (1) Inducing shock by operating a coal cutter of the "puncher" type in the working place to procure outbursts. This was impracticable as it would require an operator who would be exposed to the outburst of coal and gas, with probable loss of life. (2) Pumping water under pressure into boreholes in the face itself. This again was impracticable as the boreholes were filled with small coal immediately the drill was withdrawn. (3) Firing a volley of explosive charges in the coal seam to induce by shock the desired outburst when all the men were out of the mine.

It is said this latter method was at different times suggested to the British Columbia Mines Department, but they refused to consider it from a safety standpoint, especially as blow-outs have occurred during the change of shifts when there were no men in the working places.

RISK OF IGNITION OF FIRE DAMP

Experts agree that it would require a flame of a temperature of 1700 to 1800° F., to ignite the highest explosive mixture of fire damp with a time exposure of 0.01 second.

The makers of "Monobel," which is on the permitted list, give the temperature, from experiment, as 1590° F. with a duration of about 0.00025 sec., thereby allowing a fair margin of safety; besides, there is little chance of an explosive mixture being present on such occasion, because prior to firing, the place is free from gas.

SYSTEM OF MINING

It is important to note that the longwall system is considered the most suitable as it exposes a long line of face advancing very slowly and gives the gas a chance to expand gradually. This system was tried out at Cassidy within the last year and proved successful in eliminating the blow-outs. Unfortunately, however, the irregularity in thickness, the soft and friable nature of the coal and the number of faults and silted areas encountered make this system impracticable in the Douglas seam.

CAUSE OF BLOW-OUTS

Gas under high pressure contained in the pores of the coal undoubtedly furnishes the motive power. The compressive forces which caused

the large fold in the strata, accompanied by faulting and a movement along the coal seam, would appear to be responsible for compressing the gas.

CONCLUSION

Experience at Cassidy has effectively demonstrated: (1) That drilling bore holes in advance of working faces is of no benefit but increases the danger for the men employed on account of creating a false sense of security. (2) Placing the mine on single shift or allowing an interval of 8 hr. between shifts does not eliminate or reduce the danger of blow-outs or increase the safety of the men employed. Such restriction would result in fewer blow-outs "per month" on account of retarding development but would not in any way lessen the number of outbursts per thousand feet of development. (3) That the greatest safety for the men employed can be secured by shot-firing using permitted safety explosives under the direction of competent, licensed, shot lighters.

DISCUSSION

H. G. MOULTON, New York, N. Y.—The sections accompanying this paper show numerous, sudden changes in the thickness of the seam and the shape of the roof. There is every indication of the possibility of what may be termed geological stresses, resulting from the folding of the formation and the deformation of the seam. The structural conditions indicate that the blow-outs may be due to the sudden release of stresses in the roof through extraction of adjacent supporting coal. If the pulverization of the coal occurred as a result of release of gas inclusions in the pores, how is it possible for the mine to produce any lump coal from adjacent parts of the seam? The speaker is impressed with the thought that if gas is included in the coal, and under the pressure of the overlying strata pulverizes the coal and blows it from the face, the same gases and same pressure would exist in the coal as mined as reduce the lump to dust when brought to the surface.

G. S. RICE, Washington, D. C.—Some French engineers believe that when coal under certain conditions is under additional stress, due to the approach of workings, the total pressure becomes greater than the crushing strength of the coal causing it to break like a Prince Rupert drop, releasing instantaneously the occluded or inclosed gas which is given off then in large amounts. I am unable to accept this theory. I think the occluded gases were released by geologic local crushing ages before. Furthermore, coal mines near Alais in the south of France are subject to similar violent outbursts, but the gas given off is nearly pure carbon dioxide which could not come from the immediate crushing of the coal. I visited one of the Alais mines the day after an outburst which smothered a couple of men. The CO_2 had not yet been wholly removed by ventilation. The workings were partly filled with small coal and the finest of dust. I cannot see how the carbonic acid gas could be immediately derived from the crushed coal without being accompanied by larger amounts of hydrocarbon gases. We are carrying on experimental work on Crows Nest coal (B. C.) at the Bureau of Mines Pittsburgh Experiment Station, on the gases given off in these outbursts. The gases collected *after* the outbursts have been chiefly methane (in one instance 96.3 per cent. methane, 3.5 per cent. CO_2 , and 0.23 per cent. ethane and propane), yet when we crush a sample of coal from an outburst area in the Coal Creek Mine, it contains a large percentage of ethane, propane and other paraffin

gases. A typical analysis of gases from grinding the coal (from an outburst area) in vacuo showed 50 per cent. methane, 8 per cent. ethane, 22 per cent. carbon dioxide, and 20 per cent. nitrogen. There have been outbursts recently in South Wales, and there are reports on them in the *Proceedings* of the South Wales Institute of Engineers. I have had occasion to write a discussion on the subject.¹

T. L. McCALL, Stellarton, Nova Scotia.—This question of gas bursts is one that has given me considerable anxiety also. We occasionally have these outbursts of gas and they have caused us a lot of serious thought, but we have not yet arrived at any conclusion as to the best method of overcoming this trouble.

G. S. RICE.—I think that it has been more generally concluded in Belgium, as well as in British Columbia and France, that the advance bore holes have not been effective in draining gas, nevertheless I believe that they are of value in certain instances.

J. J. WALSH, Wilkes-Barre, Pa. (written discussion).—The author describes the manner of drilling holes in advance of the working face and states that practically no gas is evolved from these holes. Another surprising feature is the absence of any considerable volume of gas after an outburst of coal has taken place. Gas may be retained by coal in two ways: (1) The gas may be mechanically imprisoned in the pores of the coal and while so confined, it is in its natural state; (2) the gas may be occluded and when in this state it approaches a liquid form under tremendously high pressure. In order that a gas be transformed to a liquid state it must be subjected to a very high pressure. It would then seem that the gas in the mine in question is "occluded" and that the cause of the outbursts of coal is due not so much to the gas mechanically imprisoned, but chiefly to the occluded gas in the liquid or quasi-liquid state.

Occluded gases under high pressure are chiefly found in disturbed or folded areas.

R. R. WILSON (written discussion).—One of the latest theories advanced to explain the origin of the outbursts is the "electronic theory." The idea is that the electrons combine in such a way as to form instantly methane or other gas under high pressure.

In my opinion theories of origin are not nearly so important as practical suggestions to avoid loss of life and protect a company's investment as far as possible.

I would not be in favor of shotfiring from the surface as recommended by some prominent European engineers several years ago. Such procedure might cause a serious explosion which would wreck a mine. While such a method might avoid loss of life; it would introduce a great element of risk from a mine operator's point of view. With experienced, licensed shot lighters carefully testing each working plane before firing shots and all underground employees thoroughly drilled in the method of procedure, I believe the greatest possible safety, both for the men employed and the company's investment can be obtained. I would not of course recommend shotfiring where other conditions such as dust or accumulations of gas would make it dangerous.

I. TOLMACHOFF, Pittsburgh, Pa. (written discussion).—In addition to the three theories advanced for the cause of outbursts I would suggest a fourth which I regard the most probable. I do not believe that gas accumulations of any description exist in coal or have existed there for some time, but that gas is generating all the time within the faulted zone owing to the movement of strata. This is probably still going on in

¹ Discussion by G. S. Rice of "Outbursts of Gas, and Methods of Working Seams Liable to Them," *Proc. South Wales Inst. Engrs.* (Mar. 17, 1927) 43, No. 1, 32-47. For the original paper, by George Robblings, F. G. S., see (Nov. 9, 1926) 42, No. 5, 465, and other discussion, (Apr. 21, 1927) 43, No. 2, 248.

the area under consideration, producing what might be called a cold distillation of coal. The great number of recorded outbursts, enormous pressure and at the same time small amount of gas given off, as for example during the fatal outburst on April 7, 1926, strengthen, in my opinion, the theory just submitted. Comparison of analyses of coal taken so far as possible from the affected zone with those from the faulted zone, would perhaps further support the theory.

I am not inclined to consider the selected system of mining as the best one for this mine, or better to say, I do not consider the selected system correctly applied in this instance. I think it would be preferable to cross the faulted zone by a few advance tunnels, giving gas a chance to expand in different places along the faulted zone and thus replacing strong outbursts with a number of small harmless ones. Advance holes are practically aiming at the same end, but they are of no benefit owing to their small size and might even be dangerous, provoking an outburst and directing it like the barrel of a gun.

G. S. RICE (written discussion).—Mr. Walsh implies that not much gas is given off following an outburst. This was certainly true in the outburst described, but not in many previous outbursts in the Cassidy mine. In March, 1922, an outburst occurred which the fireboss reported gave off 1,200,000 cu. ft. of pure gas at the lowest estimate.

The No. 62 outburst in the Cassidy mine which occurred Aug. 5, 1922, and which I had occasion to investigate threw out or loosened 1200 tons of coal. The fireboss reported that the outburst gave off "enormous quantities" of methane. Some of the outbursts which have occurred in the Crows Nest Pass coal field have also thrown out tremendous quantities of gas. At the Carbonado mine in 1904, an outburst occurred which filled the level with coal for 400 ft. and the volume of gas given off in the first 30 min., former Chief Inspector Thomas Graham states, "was estimated by various authorities to be from 2,000,000 to 5,000,000 cu. ft. Many other instances of the discharge of large amounts of gas are reported in the annual reports of the Departments of Mines, respectively, of British Columbia and Alberta, also of France, and Belgium. In Belgium, in the period between 1847 and 1908, there were 357 instantaneous outbursts of gas causing the death of 447 miners. In some of these enormous quantities of gas were given off, notably at the Agrappe Colliery in 1879 where the air currents were reversed and 121 men were suffocated.

In the case of the Alais mines in southern France, already referred to, there have been instantaneous outbursts of carbon dioxide gas which have reached the surface and killed animals in the vicinity of the shafts.

From my observations in investigating effects of such disasters, one finds, as might be expected, all kinds of variations in the size of the outbursts, from one which may throw out only a half ton of coal or dust and release very little gas, to one of enormous size such as those cited. As far as I have been able to determine, from a review of the extensive foreign literature, as well as from observations in western Canada, there is no direct relationship between the volume of gases given off in the outbursts and the amount of coal and dust thrown out, indicating that some of this gas has been confined in extensive fault planes or joint planes adjacent to the point of outburst, in addition to that which I conceive is held exterior to the coal dust particles by surface tension. I doubt whether the normal occluded gases in large pieces play any part at the time of outburst.

In reference to Mr. Wilson's comment that he is not in favor of shotfiring from the surface when all the men are out of the mine, this method has been used extensively in the south of France for over 10 years and the French engineers consider it a successful method. It does not prevent all outbursts, as indicated by the one I investigated in that district. On the other hand, they claim that the method by the jarring and

shattering of the coal face has induced many outbursts which otherwise might have occurred when the men were in the mine. Of course, if a coal is soft enough to be worked without blasting and if brushing can be done by pick, then it may be more doubtful to introduce the blasting method. The British Mines Department permitted the experiments of blasting in the South Wales anthracite mines as described in a paper before the South Wales Institute of Engineers, already referred to. The results were said to be favorable in inducing outbursts which otherwise might have occurred when the men were in the mine.

I wish to take issue with those who have considered that boring holes in advance with the hope of draining the gas presents a serious hazard. There has been little indication of the method being serious. The British Columbia Department of Mines began requiring it, about 10 years ago, in places deemed subject to outbursts: I understand that there have been one or two near-accidents but no serious ones although thousands of holes have been drilled in advance.

If shotfiring from the surface is not practiced, I believe that advance bore holes should be used. It has been claimed by some operators in Alberta, Canada, that the method was successful. There is no way, however, of determining whether an outburst may have been prevented by such prior drainage. Perhaps in many instances the examination for gas in the holes has been perfunctory. Many of the European engineers believe that one means of preventing outbursts is to carry forward a wide face such as a longwall face. Nevertheless this has not given sufficient drainage in all cases and many outbursts have occurred in longwall faces, usually at places of folding or near faults, as shown in the annual reports of the Belgian Government.

While as Mr. Wilson says, the theories of origin are not as important as practical suggestions to avoid loss of life and property, yet unless we can fix an origin we cannot very well arrive at the best methods to be taken. I doubt if there is any one single method. So far as the conditions occur in the Crows Nest Pass mines, in which Mr. Wilson is interested, they seem to have been eminently successful in withdrawing the men from indications by sound and movement in time to prevent serious accidents. In other mining districts the geological situation is different.

In my discussion of the outbursts in the South Wales mine already referred to, I suggested that the occluded gases released ages ago by local geologic movements, were stored in the interstitial spaces of the broken coal. When coal is crushed hydrocarbon gases are given off to the amount of one to three volumes, at atmospheric pressure, of the coal and a certain amount of carbon dioxide and nitrogen are liberated.

To account for the large amount of carbon dioxide given off in some of the European outbursts, I have suggested that it was due to differential diffusion over a long period of time through the crushed but recompacted coal dust, as cases of outburst almost always occur only at depths of over 1000 ft. I have initiated some experiments which I hope to be able to report upon at some later time.

I had in mind the possibility that analyses of the samples taken from advance boreholes might disclose valuable information and I had proposed such sampling in a certain mine subject to outbursts. Recently my attention has been directed to a German report which appeared in *Gluckauf*.² The authors claim that when the percentage of CO₂ obtained by evacuating the gases from the advance bore holes rises above 40 per cent. an outburst is liable to occur and when it goes over 84 per cent. one is almost certain of occurrence.

² C. Kindermann and L. Tolsdorf: Die Untersuchung der Bohrlochgase als Mitte; zur Vorauserkennung von Gasausbrüchen untertage. *Gluckauf* (Oct. 30, 1926) 1441-1444.

Methane Content of Coal-mine Air*

By W. P. YANT† AND L. B. BERGER,‡ PITTSBURGH, PA.

(New York Meeting, February, 1927)

THIS paper presents evidence of the general occurrence of methane in a large number of the coal fields of the United States and substantiates the apparent unnecessary differences in the ventilation requirements and codes in the various states, as was brought to the attention of the Committee on Mine Ventilation of the American Institute of Mining and Metallurgical Engineers at its Pittsburgh meeting in October, 1925, by J. A. Garcia.¹

The data in his report have been compiled from the files of the Gas Laboratory of the Pittsburgh Experiment Station of the Bureau of Mines and represent samples collected throughout the various states and submitted for analysis by its engineers, also by mine operators and State mining departments, since 1911. Several thousand analyses were available but most of them were rejected because they represented abnormal conditions, undefined place of sampling, special investigations, or duplications in the same mine. Only those samples were included which, without reasonable doubt, were ascertained to fall within three major classes, namely, main return, air splits, and face and rooms of live workings. In the case of air splits, and face and room samples, only the one which contained the highest percentage of methane found in a given mine was taken.

OCCURRENCE OF METHANE IN COAL MINES OF THE UNITED STATES

Obviously it is not feasible to present the detailed analytical data here and accordingly these data have been compiled into groups in a manner which it was thought would bring out clearly their significance. These are shown in Table 1 which includes the analyses of 310 samples of return air of approximately the same number of coal mines in 19 states. Because of changes in ownership and names the exact number could not be ascertained. The table also includes 337 samples from air splits and 263 from the face and rooms of some of the same mines represented by return air samples and from other mines in these states. A total of more than 400 mines are represented by Table 1.

* Published with approval of the Director, U. S. Bureau of Mines.

† Supervising Chemist, Health Laboratory Section, Pittsburgh Experiment Station.

‡ Laboratory Assistant, Gas Laboratory, Pittsburgh Experiment Station.

¹ J. A. Garcia: State Coal Mining Laws. *Trans.* (1926) 74, 409.

TABLE 1.—*Distribution of Methane in Coal Mines of Various States*

State	Methane, Per Cent.								Total Number Samples
	0.00 0.05	0.06 0.10	0.11 0.25	0.26 0.50	0.51 1.00	1.01 2.00	2.01 5.00	5.01 and Over	
Alabama									
Main return.....	15	5	17	6	2				45
Air splits.....	2	4	15	4	6				31
Room and face.....	3		7	4	8	2	5	3	32
Colorado									
Main return.....	13	4	6	6	2	2			33
Air splits.....	2	2		3	2	1			10
Room and face.....	6	2	5	2	4	2	4		25
Illinois									
Main return.....	3	3	8	2	1				17
Air splits.....	37	35	71	21	4	1			169
Room and face.....	1	1	2	3	4	3		7	21
Indiana									
Main return.....	1	3	4	1	2	1			12
Air splits.....	10	2	15	16	4				47
Room and face.....	1	1	4	2	3	3	3	6	23
New Mexico									
Main return.....	26	8	3	3	1	1			42
Air splits.....	1					1	1		3
Room and face.....	8	8	3	1	1	2			23
Pennsylvania									
Main return.....	10	9	14	6	4	5			48
Air splits.....	1	8	4	4	6	1	2		26
Room and face.....	10	3	4	3	6	6	7	11	50
West Virginia									
Main return.....	12	8	3	4	3	1			31
Air splits.....	3	2	7	10	1	2			25
Room and face.....	6	3		2	3	8	3	7	32
Wyoming									
Main return.....	14	3	2	2	1				22
Air splits.....	2	1			2				5
Room and face.....	7	1	3	2	1	1	1	3	19
Arizona									
Main return.....	2			1					3
Air splits.....									0
Room and face.....									0
Arkansas									
Main return.....			1	1					2
Air splits.....									0
Room and face.....	1				2		1		4
Kansas									
Main return.....		3	2						5
Air splits.....			4						4
Room and face.....			1		1			1	3
Kentucky									
Main return.....	8	3	4						15
Air splits.....	1	3		1	4				9
Room and face.....	2		3	1					6
Maryland									
Main return.....	2								2
Air splits.....									0
Room and face.....	1				1				2

TABLE 1.—*Distribution of Methane in Approximately 300 Coal Mines of Various States.—(Continued)*

State	Methane, Per Cent.								Total Number Samples
	0.00 0.05	0.06 0.10	0.11 0.25	0.26 0.50	0.51 1.00	1.01 2.00	2.01 5.00	5.01 and Over	
Montana									
Main return.....	2								2
Air splits.....				1					0
Room and face.....									1
North Carolina									
Main return.....				1	1				2
Air splits.....					1				0
Room and face.....									1
Ohio									
Main return.....	2		2	1					5
Air splits.....	2	1	1						4
Room and face.....		1		1		1		1	4
Oklahoma									
Main return.....	1	1	2	2	2	1			9
Air splits.....				1		2			3
Room and face.....					3	1	1	6	11
Tennessee									
Main return.....		1	3						4
Air splits.....									0
Room and face.....								1	1
Utah									
Main return.....	5	1	5						11
Air splits.....			1						1
Room and face.....	1	1	2	1					5

DISCUSSION OF DATA PRESENTED

It is obvious from these data that gassy mines exist in every state listed. While it is true that the number of samples representing some of the states as Arkansas, Arizona, Kansas, Maryland, Montana, North Carolina and Virginia are too few to show adequately the wide occurrence of methane, the data given were the only results available which were in the classes designated. However, analyses of many samples from other locations in mines in these states were available, and if the results were included they would show gassy mines to exist in virtually every coal-mining state.

Methane and Explosions

This wide occurrence of methane in dangerous quantities is further emphasized by the fact that disastrous explosions, caused by either methane alone or methane and coal dust, have occurred in almost every coal-mining state, and in general they have been distributed in number in proportion to the magnitude of the coal industry in the respective states.

Pennsylvania leads the list of explosions by a good margin, followed by West Virginia, Oklahoma, Illinois, Indiana and Alabama as close competitors for second place, and Colorado, Kentucky and Kansas for third. This apportionment is based on the record of explosions at which the Bureau of Mines has been requested to give assistance. There have been many additional explosions but it was thought that they will not materially change the above statement. All of the explosions recorded have virtually had a uniformity of causes giving rise to the accumulation of methane and likewise the causes of ignition have been quite similar.

It thus appears that because of the general occurrence of methane, similar problems exist in nearly every state, yet wide differences exist in regulations pertinent to ventilation. Mr. Garcia attributes these differences in the regulations to the fact that they have been developed over a long period by innumerable individuals and legislatures, and without cooperation between states or any attempt to codify them.

In view of the trend of the industry toward improved methods in general, and also the increased information regarding the factors conducive to safe and unsafe mining practices, it appears that the time is opportune for a revision of many of the sections of mining codes. As specific examples might be mentioned the placing of ventilation standards on the basis of composition of the air rather than on the present basis of cubic feet of air per minute per man, for in the final analysis the adequacy of ventilation with respect to fire damp is measured in terms of flammability either actual or potential. For this reason as well as from the standpoint of health, it appears that a logical and fair standard should be based as far as possible on the composition of the atmosphere.

The inefficiency of the basis of volume of air per man, and the need for a regulation based on composition, appears to be manifest in Table 2 which gives the average composition of 272 samples of air collected from the main return of about a similar number of mines. These samples have been arranged in groups on the basis of their methane contents, similar to that given in Table 1 as 0.00 to 0.05, 0.06 to 0.10, etc., per cent., and the values given for each group are the average analyses for methane, carbon dioxide and oxygen, respectively.

TABLE 2.—Average Analyses (Per Cent. by Volume) of 272 Samples of Return Air for Methane, Carbon Dioxide and Oxygen

Group.....	0.00 to 0.05	0.06 to 0.10	0.11 to 0.25	0.26 to 0.50	0.51 to 1.00	1.01 to 2.00
Number of samples.....	100	45	69	33	17	8
Percentage of samples.....	36.8	16.6	25.4	12.1	6.2	2.9
Gas:						
Methane, per cent.....	0.02	0.09	0.16	0.37	0.69	1.26
Carbon dioxide, per cent....	0.13	0.13	0.24	0.20	0.17	0.14
Oxygen, per cent.....	20.71	20.70	20.53	20.51	20.43	20.36

If we regard the diminution in the oxygen content and the increase in carbon dioxide, which are due mainly to the reactions attending absorption of oxygen by the coal, as a general index of the quantity of air it appears from Table 2 that the quantity of effective air was not increased in proportion to the methane liberated. The amount of carbon dioxide in the return air is almost the same, regardless of the increasing methane content. The oxygen, however, shows a gradual decrease, but on the average this is but little if any more than would be caused by the dilution with methane. One per cent. of methane introduced into air will reduce the oxygen content about 0.2 per cent. The outstanding significance of this rather indirect evidence is the lack of a definite connection between the amount of methane and the quantity of air.

In connection with the results shown on Table 2 it is interesting to note the close agreement between the average of all these data excepting the last column with the average analyses of samples of return air from 100 mines which was published about 10 years ago by the Bureau of Mines.² These analyses are given in Table 3.

TABLE 3.—*Comparison of Average Analyses (Per Cent. by Volume) of Return Air from Mines*

Source	Methane, Per Cent.	Carbon Dioxide, Per Cent.	Oxygen, Per Cent.
Data from 272 mines as given in Table 2. . . .	0.17	0.17	20.54
Data from 100 mines compiled 10 years ago..	0.20	0.18	20.51

CLASSIFICATION OF COAL MINES WITH RESPECT TO METHANE

The difference of opinion among states and the lack of a recognized basis and procedure for classifying coal mines with respect to actual or potential hazards from methane—that is to say, gassy or non-gassy mines—parallels the case of ventilation standards, and for the same reasons. It is unnecessary to discuss such phases as the inadequacy of the flame safety lamp as a sole basis of classification; the inadequacy of a mere analysis of the return air; the variability if not often fallibility of judgment when left to the discretion of different individuals; or the absurd and misleading if not dangerous practice of applying the term non-gassy to a class of mines for which the definition permits a certain amount of gas. The general matter of consideration at this time is that the present situation is apparently undesirable, not only because it does not give a definite and adequate means for ascertaining the class to

² G. A. Burrell: Properties of Mine Gases. Bur. Mines Chart.

which a mine belongs, but also because it does not promote the seemingly desirable standardization of terms and specifications.

SUMMARY

In this report which deals with the composition of approximately 900 samples of coal mine air collected in coal mines in the United States since 1911, it has been desired to present information which was thought to be of interest and pertinent to ventilation problems of the coal mining industry. The wide occurrence of gassy mines and the apparent need of a more standardized ventilation code are pointed out.

ADDENDUM

Since this paper was presented the authors have noted an abstract³ of an article by A. Girard in *Revue de l'Industrie Minerale*, in which are described the results of measurement of methane in the return air of coal mines of the Sarre basin. At the end of 1925 the composition of the air was as follows:

Methane, Per Cent.	Air Returns Showing the Percentages of Methane	
	Number	Per Cent.
0.01 to 0.10	19	27.9
0.11 to 0.50	36	53.0
0.51 to 0.75	9	13.2
0.76 to 1.00	3	4.4
1.01 to 1.25	1	1.5

While a comparison of the results of Table 2 which represents samples collected from 272 coal mines in various parts of the United States since 1911, with the results given by A. Girard for the Sarre basin at the end of 1925 is not fully justified, it is interesting to note that since 1911 the number or percentage in Table 2 which contains 0.10 per cent. or less methane is approximately twice that given for the Sarre basin, but that the total of all mines having less than 0.5 per cent. methane in the return is 91 and 81 per cent., respectively, or approximately the same for both places.

A. Girard states that explosions are fairly frequent in this section. Also, that the number of stations for testing air for methane have been markedly increased which indicates a wider application of gas analysis to problems in coal-mine ventilation.

In preparing the present paper it was in a large measure the authors' purpose to stimulate a wider application of gas analysis to mine ventilation problems in this country.

³ Gas Testing in the Sarre Mines. *Colliery Guardian* (Sept. 2, 1927)•135, No. 3479, 49.

DISCUSSION

F. HAAS, Fairmont, W. Va.—I believe this is the first time this subject has been put before the Institute. My criticism is that the writer has confused the question of ventilation with the quantity of gas the mine generates, resulting in an indefinite answer to either.

The authors' information is based on occasional samples. My experience indicates that there is considerable variation both in ventilation and quantity of gas liberated and conclusions drawn from a single or occasional samples are misleading.

R. D. HALL, New York, N. Y.—I should like to know how you chose those particular samples. On what basis did you divide them up among the various mines in the country?

W. P. YANT.—I went through our files and tabulated the results of analysis of all samples which were of the three types described. No samples which did not fall in these groups were taken. On the other hand, however, I included every sample that did.

R. D. HALL.—On what basis were they originally collected, every mine in the country?

W. P. YANT.—These samples are of the three types described that have been sent into the laboratory for analysis since 1911. Some were taken by the Bureau's engineers, some by operators, and some by state departments. They do not however represent every mine in the country. As stated, they were taken in approximately 300 mines in various states.

R. D. HALL.—More or less by chance?

W. P. YANT.—Yes. I agree with Mr. Haas regarding the use of data of this kind. I merely tabulated the analyses, to show what the conditions are throughout the country and to point out certain trends, not conclusions. I could have included samples, which were taken periodically in a number of mines over a period of several years, as well as I could have included air quantities, but I could not have incorporated all these in one short paper.

G. S. RICE, Washington, D. C.—I think the purpose of Mr. Yant is primarily to indicate the trend of ventilation in gassy mines. If in future he further studies this subject, I suggest including the volumes of the corresponding ventilating currents. Coal mines generally throughout the country are becoming deeper and deeper and increasing quantities of methane are encountered, but there have also been material increases in the volume of air put into the mine. When further compilations are made, if the quantities of air were included the figures in conjunction with percentages of methane, would indicate the degree of improvement in ventilating methods.

Devices for Detecting Dangerous Gases in Mine Air

BY JOHN T. RYAN,* PITTSBURGH, PA.

(New York Meeting, February, 1927)

SIR HUMPHRY DAVY's epoch-making treatise delivered on Nov. 9, 1815, before the Philosophical Society of London, first announced and demonstrated a flame safety lamp for detecting methane in mine air. Coal mining was known in England as early as 1180, more than 600 years before the advent of a safe lighting and gas-detecting appliance. Mining, no doubt, took place first in outcrop coal, and danger from gas did not become apparent until the workings became deeper and more extensive. Davy's lamp did much to increase the safety of miners, and a modified Davy lamp is still widely used for detecting the presence and approximate amount of combustible gas in mine air.

No factor in coal mining has improved more rapidly during the past 15 years than safety measures. This progress came about not only as a result of the workmen's compensation laws now in force in most of the states of the Union, but because the research work of the U. S. Bureau of Mines had demonstrated some dangers incident to coal mining which were not considered to be serious matters prior to that time.

Electric power and explosives are necessary but potential hazards in American coal mines. A general mechanization program is now under way, while the mines are growing deeper and more extensive. Consequently ventilation problems are more complex and a demand has been created for positive detecting devices for carbon monoxide as well as methane. Scientists and manufacturers, working hand in hand during the past 10 years, have invented and developed appliances which make systematic and effective examination of mine air possible. The object of this paper is to describe briefly the approved testing devices in use today, with the hope that they may lead to the saving of life.

TWO GROUPS OF DANGEROUS COAL MINE GASES

Dangerous coal mine gases may be placed in two groups: (1) those that are dangerous because of their explosive properties, and (2) those that are dangerous because of their toxic properties. Methane (CH_4) is the important gas of the first group, and in so-called gaseous mines constantly exudes from the coal and rock; it may accumulate and form an explosive mixture.

The explosive range of methane-air mixtures is well defined. The low limit is 5 per cent., the high limit 15 per cent., and the most explosive

* Vice-president and general manager, Mine Safety Appliances Co.

mixture is about 10 per cent. methane. Any amount of methane in mine air increases the inflammability of coal dust, and one-half of 1 per cent. will materially increase the rate of propagation of coal-dust explosions. Hydrogen gas (H_2) should be mentioned, but it is mostly found in fire areas.

The common toxic gas is carbon monoxide (CO), a product of incomplete combustion that is always associated with mine fires, and is one of the principal constituents of "afterdamp" following explosions. Carbon monoxide is also explosive, but the lower limit is about 12.5 per cent. in normal air. When this percentage of carbon monoxide is present in mine

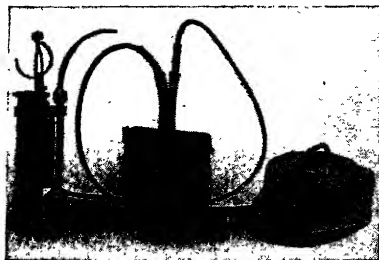


FIG. 1.—BURRELL PERMISSIBLE METHANE INDICATOR.

air, the oxygen (O) content is not sufficient to support an explosion. On the other hand, carbon monoxide is highly toxic, and an explosive mixture would cause almost immediate death to a person breathing it. The presence of carbon monoxide in mine air may also be associated with spontaneous oxidation of coal, and its presence above certain normal values must be considered as an absolute indication that heating of some kind is taking place in some part of the mine. Oxides of nitrogen and hydrogen sulfide (H_2S) are seldom found in dangerous quantities in coal mines, but they do occur in some metal mines.

ACTION OF CARBON MONOXIDE ON HUMAN BEINGS

Carbon monoxide has an action on human beings which is peculiar to itself. When inhaled it comes into contact with the blood brought to the lungs and it unites chemically with the hemoglobin, the oxygen-carrying constituent of the blood. After this chemical combination has taken place the individual who has inhaled the gas is deficient in oxygen, through impairment of oxygen-carrying capacity of the blood, just as much as though the same amount had been lost by the opening of one of the blood vessels or his supply of air had been shut off. A few breaths of air containing 2 per cent. of carbon monoxide will cause unconsciousness and death in 3 or 4 min. Death is likely to occur when a man

breathes air containing as low as 0.2 per cent. of carbon monoxide for from 2 to 4 hr. As a matter of fact, relatively long exposures to any percentage above 0.01 per cent. must be regarded as unhealthful or dangerous, and particularly during vigorous exertion, when the volume of breathing is greatly increased, and consequently saturation of the blood with carbon monoxide is more rapid. The Bureau of Mines has recently defined the permissible limits of carbon monoxide in vehicular tunnels as 0.04 per cent. for 1 hr., and in mines at 0.01 per cent.

Carbon dioxide (CO_2), oxygen deficiency, and humidity sometimes seriously complicate ventilating problems, but will not be discussed at any length in this paper.

GASSY AND NON-GASSY COAL MINES

From a practical point of view all coal mines are potentially gassy. However, coal mines are now designated by the various states as gassy

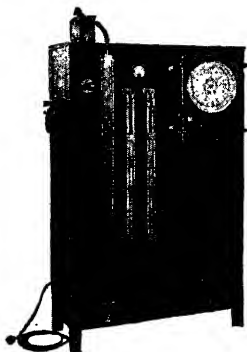


FIG. 2.—METHANE CONTINUOUS RECORDER.

or non-gassy for purposes of regulation in respect to the prevention of explosions and fires. For example, the Pennsylvania bituminous mine law provides in general that any mine which generates explosive gas in a quantity sufficient to be detected with an approved safety lamp is a gassy mine. This does not apply to mines generating explosive gas only in live entries. Furthermore, if no explosive gas is detected by means of an approved safety lamp within one year a mine may be termed non-gassy, according to the judgment of the mine inspector.

In gassy mines, and some so-called non-gassy mines, the standard practice is to examine the entries and working places before miners are permitted to enter. The examination is made by "fire-bosses" who make their runs several hours before the regular shift begins, and examine a prescribed area with a flame safety lamp. A few mining companies employ gas inspectors to check the fire-boss examinations daily by samp-

ling and analyzing the return air, and one or two companies sample the air of each return split as well as the main return. In a number of cases the Burrell indicator is used by foremen in examining the return from live sections of the mine.

DETECTION OF GAS

Fire-bosses generally use a luminous flame in testing for gas. Some claim that they do not have the time to visit all places on their section without hurrying, and cannot take the time to make a more accurate non-luminous flame test. Others claim that gas can be more easily detected by the "pulling up" of a normal luminous flame than by the gas-capping of a non-luminous flame. Safety-lamp testing at best is only roughly quantitative, and the luminous flame method is unreliable. If the luminous flame is not pulled up suddenly by the gas it is practically impossible to tell whether gas is present or not. If a lamp is moved suddenly into the atmosphere to be tested, and then is stopped abruptly, the flame may lengthen although there is no gas present; on the other hand, if the lamp is slowly introduced into the atmosphere to be tested the flame may be pulled up too slowly to be discernible.

The Bureau of Mines states that the length of the gas cap formed over a non-luminous flame of a Wolf roundwick lamp when 1 per cent. of gas is present is about 0.3 in., and many men claim that they can easily and accurately detect this percentage. This is doubtful, because it is easy to be misled unless one is extremely careful to differentiate between the "fuel" cap, which is constantly formed, and the gas cap. It is doubtful, considering all the variables, whether less than 2 per cent. of methane can be detected effectively. This percentage gives a cap about 0.4 in. high.

The Bureau of Mines has recently offered a new classification for coal mines depending on the percentage of methane generated, and has termed a mine virtually non-gassy in which inflammable gas in excess of 0.05 per cent. cannot be found by systematic search. A definitely defined classification will no doubt meet with general approval and is likely to be standardized and accepted by the various mining states.

Methane is a constant source of worry in coal-mining operations. The majority of mine explosions have been initiated by ignition of gas, and many explosions have occurred in so-called non-gassy mines. Mine operators all over the country are on the lookout for new inventions and improved testing appliances that may be utilized by the men whose lives are devoted to coal mining.

The flame safety lamp is also useful in detecting oxygen deficiency, the flame being extinguished when oxygen decreases to about 17 per cent. In metal mines the candle will serve the same purpose. A carbide lamp should not be used for the purpose of testing for oxygen, as it is not

extinguished until the oxygen falls to about 12 per cent., and the lessening in brilliancy of the flame might be supposed to be due to other causes. The Haldane indicator is sometimes used in non-gassy metal mines for examining mine air, and it will show the percentage of oxygen present through a range of 21 per cent. oxygen down to 17, together with corresponding percentages of carbon dioxide. The gas-analysis apparatus will, of course, determine accurately the percentages of carbon dioxide, oxygen, carbon monoxide, hydrogen, and nitrogen (N_2). The sling psychrometer is used almost exclusively for measuring the relative humidity of mine air.

The selection of an appliance to detect and determine the percentage of methane in mine air depends primarily on the following factors: Lower limit of detection, accuracy, desirability of sampling and analyzing in the mine, operator, examination required for other gases, speed of manipulation, and continuous or intermittent record.

All of the devices, excepting the flame safety lamp, depend primarily on burning the gas out of the sample; the decrease in volume being an indication of the percentage of methane present. Some of the appliances classified above are too well known to need lengthy explanations, but the newer devices, together with their field of use, will be described in greater detail.

FLAME SAFETY LAMP

The capping of a non-luminous flame is a physical phenomenon caused by the burning of the methane in the air. The principal function of a flame safety lamp, since the advent of the electric cap lamp, is to indicate



FIG. 3.—CARBON MONOXIDE DETECTOR.

the presence of methane in mine air. Examinations should be made by all underground officials, as well as by fire-bosses, and sufficient time should be taken to make an effective test. The machine runner is also

required to carry a flame safety lamp, and he should be as well versed in operating it as a fire-boss. Safety lamps must be kept in perfect condition and should be charged with the best obtainable fuel. All lamps should be kept and repaired in the lamp house, which should be in charge of a reliable, well paid employee. A full size round or flat-wick lamp is the best lamp for general inspection, and most fire-bosses prefer an aluminum lamp because it is lighter in weight. Foremen, who carry an approved electric cap lamp, sometimes carry a permissible baby Wolf, which weighs only about $\frac{1}{2}$ pound.

BURRELL PERMISSIBLE METHANE INDICATOR

The Burrell gas indicator (Fig. 1) was designed in 1915 by G. A. Burrell, then chief gas chemist of the Bureau of Mines, and is used to detect smaller percentages of methane in mine air than can be detected by means of a safety lamp. The instrument will detect as low as 0.1 per cent. of methane. It consists essentially of a "U" tube, one side of which is glass and the other brass. A scale, graduated to read percentages of methane, is attached to the glass tube. On top of the brass tube is a valve through which a sample of mine air is drawn, and in the tube is a small coil of platinum wire that has terminal connections on the top of the tube.



FIG. 4.—PYROTANNIC DETECTOR.

When used the instrument is filled with water to a mark. A sample of mine air is drawn into the instrument at any point in the mine as desired, and the current from the battery of an electric cap lamp is then switched on. The current causes the platinum coil to be raised to a high temperature, sufficient to burn the methane in the sample. If methane is present there will be a decrease in volume of the air sample after the burning and cooling, and the water will rise in the brass tube, causing a much greater drop in the water level of the smaller glass tube.

The glass tube is calibrated in terms of methane and results are obtained direct from the scale.

The Burrell indicator is simple, and can easily be used by anyone after a few determinations. Only 5 min. are required to make a complete test, and while it is not adapted for the work of the fire-boss in making his regular run, it is of value to the foreman and assistant foreman in checking up the ventilation throughout the day. With its aid they can keep careful watch over the percentage of methane in each split of air, and thereby regulate the quality of air going to the splits with the exactness required to keep the percentage of methane below a certain amount.

GAS-ANALYSIS APPARATUS

Three types of gas-analysis apparatus are in use for analyzing mine air for methane: Portable methane-analysis apparatus, portable gas-analysis apparatus, and laboratory gas-analysis apparatus.

The first is designed for mine use, and will measure as low as 0.1 per cent. of methane. The portable gas-analysis apparatus will analyze mine air for all common gases, such as methane, carbon monoxide, carbon dioxide, oxygen and nitrogen. It will measure 0.2 per cent. of carbon monoxide as well as methane, and is likewise effective for the other gases mentioned. It can be taken underground when fires occur, set up and operated in the mine foreman's office, for analyses other than methane. The laboratory gas-analysis apparatus is not portable, but a skillful manipulator can determine as low as 0.02 per cent. of methane, carbon monoxide, and the other common mine gases.

A perfected methane recorder (Fig. 2) was shown for the first time at the American Mining Congress meeting in Cincinnati, May 24 to 28, 1926, and is epoch-making in respect to coal-mine ventilation. It provides an accurate, continuous record of the amount of methane in any current or split of air, and serves as a guide for adjusting regulators or the speed of the fan so that the volume of air supplied to any split may be sufficient to maintain proper ventilating standards.

An electrically-driven pump draws the air sample into the instrument. The sample first passes through a purifying chamber, which removes dust and solid particles, and then through a canister filled with chemicals for removing any moisture or carbon dioxide which may be present. The sample next passes through a temperature equalizer and then a flow meter which maintains a constant head of pressure in forcing the sample through an orifice 0.014 in. in diameter. The pressure is regulated to force a $\frac{1}{2}$ -liter sample through the orifice per minute, any excess being by-passed to the outside air. This $\frac{1}{2}$ liter then flows through a fused-silica tube containing platinized asbestos, which is heated to a temperature of about 800° C. by means of an electric furnace. The combined effect of the temperature and catalytic action of the platinized asbestos serves to

oxidize completely any methane which is present to carbon dioxide and water. These products are absorbed when the sample passes through a second canister.

Thus, by removing the products of combustion of the methane there is a contraction in the volume of the sample equivalent to three times the volume of the methane originally present. The remainder of the sample is led through the temperature equalizer which brings it back to the original temperature, and is then forced through a second orifice of exactly the same diameter as the first one (0.014 in.). If no methane is present the volume will remain constant, and the head of pressure required to force it through the second orifice will be identical with that of the first orifice. If methane is present the pressure required to force the reduced volume of the sample through the second orifice is correspondingly

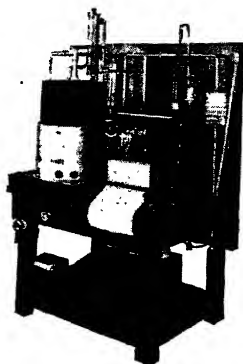


FIG. 5.—CARBON MONOXIDE CONTINUOUS RECORDER.

lower. This difference in pressure is recorded by means of a sensitive pressure recorder on a chart calibrated to read percentages of methane direct. The chart covers a range of from zero to 2 per cent. methane, and can be accurately read to 0.01 per cent. The chart shows the exact amount of methane present in the ventilating current at any minute of the day. The chart is renewed and filed daily, while the absorbing chemicals in the two canisters are renewed weekly. No other attention is normally required to keep the instrument in continuous service. This same principle can be applied to give a continuous record of such gases as O_2 , CO , CO_2 and SO_2 , with a sensitivity of 0.01 to 0.02 per cent.

CARBON MONOXIDE

The factors involved in the selection of a device for detecting carbon monoxide in mine air are quite similar to the factors previously enumerated. Dangerous quantities of carbon monoxide are produced in

mines during mine fires or following mine explosions, and small amounts may be found in mine air, due to blasting and the oxidation of coal. With the exception of the canary bird, the devices for detecting carbon monoxide enumerated in the classification depend, in the order named, on the reducing action of carbon monoxide; the affinity of carbon monoxide for the hemoglobin of the blood; contraction in volume due to combustion or absorption; and the heat liberated incident to the oxidation of carbon monoxide.

The use of canary birds in detecting carbon monoxide in mine air depends on their sensitivity to carbon monoxide. A canary bird breathes more rapidly and has a greater circulating rate than man and, consequently, absorbs carbon monoxide faster than man during the same time of exposure. In air containing 0.2 per cent. of carbon monoxide a bird will show pronounced symptoms of distress in $1\frac{1}{2}$ min. In 3 min. it becomes unsteady, and will fall from its perch in 5 min. As indicated in the classification, a canary bird is merely a warning agent of the presence of carbon monoxide in dangerous amount. Man is generally safe in an atmosphere in which a canary bird will not be overcome, but a bird will not detect extremely low percentages of carbon monoxide that will impair health if breathed for long periods. Canaries are delicate and hard to keep in rescue stations for emergency use; also, undue exposure underground often proves fatal to them.

The carbon monoxide detector (Fig. 3) was designed in 1921 to determine the presence of carbon monoxide in dangerous proportions. The operation of the device is very simple and it is now considered a standard mine-rescue and fire-fighting appliance. The detector tube used with the instrument is filled with a chemical which changes color when air containing carbon monoxide is passed through it. The shade of the color (bluish-green) varies with the concentration of carbon monoxide. By comparing this with the standard color scale, which is included with the instrument, not only the presence but the approximate amount of carbon monoxide in the air under test can be quickly determined. About 0.07 per cent. can be easily detected with the device. The barrel of the instrument is filled with activated charcoal, which absorbs any other gases in the atmosphere that might affect the chemical in the detector tube. Ten squeezes of the bulb are all that is needed to make a test. The color produced in the detector tube soon fades upon standing, permitting its re-use for seven or eight ordinary tests, when it should be replaced with a new tube.

The advantages of the carbon monoxide detector over the canary in mine-rescue work are: It will detect a smaller percentage, it gives almost immediate results, and the operator knows at all times whether CO is present and approximately the percentage. When 0.2 per cent. CO is encountered by a rescue crew wearing breathing apparatus or approved

gas masks the canary is no longer available and the crew either has to return with it to fresh air or it dies and, in either case, is no longer available for advance exploration.

The pyrotannic method of detection is based on the same principle and reactions as those which take place when carbon monoxide poisoning occurs—namely, the combination of carbon monoxide with hemoglobin of the blood. Hence it may be regarded as a specific test. The blood examined may be either that of a victim of supposed carbon monoxide poisoning, in order to verify the diagnosis, or carbon monoxide may be detected in the air by collecting a sample of the air and introducing a small amount of non-poisoned blood solution into the sample container. In the latter case the combination of CO and hemoglobin will take place in a manner similar to that which has been described as taking place in the lungs, and an examination of this blood will reveal whether or not carbon monoxide was present in the air to which it was exposed.

PYROTANNIC DETECTOR

The detection of carbon monoxide by this method is based on the fact that a grayish brown suspension will form in a few minutes after treatment of normal blood solution with a mixture of tannic and pyrogalllic acid, whereas when the blood is combined with carbon monoxide it will be faint to deep red, depending on the degree of poisoning. By preparing color standards to represent certain degrees of poisoning, the unknown samples of blood may be matched to the color of these standards and the amount of carbon monoxide ascertained. Only a small amount of blood solution, 0.10 c. c., or 2 or 3 drops of blood diluted to 2.0 c. c. with water, is required for an analysis, the blood being obtained by pricking the finger slightly with a small spring lancet furnished with the apparatus. In cases of supposed poisoning, the blood solution is transferred to a small test tube and treated with the tannic and pyrogalllic acids.

The procedure for examining mine air by this device is to collect approximately 250 c. c., or $\frac{1}{2}$ pint, sample of gas, introduce 2.0 c. c. of the blood solution into the bottle, close the latter and agitate by rotation for about 10 min. During this agitation the reaction between the hemoglobin of the blood and the carbon monoxide will take place, after which blood solution is drained into a small test-tube, and a small amount of the tannic and pyrogalllic acid added. The tube is inverted a few times to mix the blood and acids after which it is allowed to stand 15 min. and comparison made with the standards.

This device may be used to determine accurately carbon monoxide from 0.01 to 0.20 per cent. Thus it covers the range of concentrations from those which produce no appreciable effect after several hours, to those which will cause headache, nausea, dizziness, collapse and death,

and for these reasons is well suited to health investigations. Amounts of CO higher than 0.2 will be indicated as 0.20, or a dangerous atmosphere.

In addition to its use for health problems, the device is very useful for determining accurately the presence or absence of carbon monoxide in air collected from behind fire seals, thus giving information regarding the condition of the fire. These traces, which are very significant, are difficult to determine with the desired accuracy by the usual methods of gas analysis.

Probably the greatest contribution in recent years in the field of devices for detection of dangerous gases has been the CO recorder, which was perfected and placed on the market in 1925. Its development has involved so much research work that the entire history of it should be recorded in this paper.

PRINCIPLE OF OPERATION OF CO RECORDER

Its fundamental principle is based on the measuring of heat developed in the catalytic oxidation of carbon monoxide by "Hopcalite." The heat liberated in this oxidizing process is directly proportional to the amount of carbon monoxide present and is measured by a thermometer or a unit of thermocouples wired to a recording potentiometer. The chart of the potentiometer is calibrated to read carbon monoxide direct with a 0.0002 per cent., or 2 parts per 1,000,000 of carbon monoxide. The apparatus in its present form represents the combination or assembly of various ideas and devices developed through a period of 8 years to meet the demands of war and industrial gas emergencies.

The earlier contributors were Doctors Frazer and Scalione and their co-workers in the Chemical Warfare Service during the war, who developed Hopcalite to afford protection against carbon monoxide gas. There was an urgent war need also at that time for a carbon monoxide indicator, and Doctors Lamb and Larson, also of the Chemical Warfare Service, worked on the problem, developing and later patenting what is substantially in principle the present commercial recorder, except that other catalysts were used, as Hopcalite was not then available.

IMPROVED FORM OF CO RECORDER

At the end of the war these important developments, all of which had been patented, became available for use in providing protection against the increasing hazard of carbon monoxide in industry and particularly in our mines. In 1919 work was started on the further perfection of Hopcalite for use in gas-mask canisters to provide protection against carbon monoxide at all breathable temperatures. This work was under the direction of R. P. Mase, a research chemist. During the conduct of this work it was very necessary to know accurately the percentages of CO

entering and leaving the canister, so fulfilling the old adage that "necessity is the mother of invention," he combined the Lamb-Larson development with Hopcalite, which had now been perfected, and added to these his own invention of the Mase compensating flowmeter, working out what was termed, for lack of a better name, a "visible CO indicator."

This provides a means of drying the air under test and purifying it of substances other than carbon monoxide, of securing a uniform rate of flow of the sample through the compensating flowmeter, and of then passing the sample through a chamber of Hopcalite maintained at a uniform temperature by immersion in boiling water. A sensitive thermometer is placed in the Hopcalite so that any rise in temperature due to oxidation of carbon monoxide can be seen at a glance. A scale accompanying the thermometer shows the corresponding percentages of carbon monoxide. This indicator has been in service since early in 1920 and will accurately indicate concentrations of carbon monoxide as low as 10 parts in 1,000,000.

With the advent of vehicular tunnels, such as the Liberty tunnels in Pittsburgh, and the Holland tunnels under the Hudson river, connecting New York with Jersey City, it was realized that adequate ventilation would have to be provided to take care of the carbon monoxide from the exhaust gases of automobiles. The U. S. Bureau of Mines cooperated in making the ventilation studies. In this work they required an instrument which would continuously and accurately record small percentages of carbon monoxide. The development of such a recorder was begun in 1923.

The Bureau of Mines accordingly started with the visible indicator as the basis, substituting, in place of the thermometer, a unit of thermocouples and wiring these to a Leeds and Northrup recording potentiometer. This, of course, involved a great deal of work in determining the best operating conditions and general arrangement of the apparatus. The recorder was perfected by the Bureau of Mines to a point where it will show carbon monoxide in concentrations as low as 2 parts in 1,000,000. The recorder is described in the U. S. Bureau of Mines *Technical Paper No. 355* and is now commercially available. The instrument has special application in mining operations for the detection of spontaneous combustion or mine fires in their incipient stage. It can be installed on the main air return.

Mine gases are no longer considered as "inherent" dangers in mining operations. Without present knowledge, based on reliable experiments, methods of guarding against mine explosions and asphyxiation have been resolved into rather systematic procedures. Methane and carbon monoxide are dangerous gases and disaster is still a penalty for breaking any of Nature's laws in respect to these gases. A blind trust in fatalism will not prevent the exaction of that penalty. The various appliances discussed in this paper apply directly to ventilating problems, and if used

intelligently will do much to eliminate explosions and asphyxiations. The writer wishes to acknowledge valuable suggestions, received from Edward Steidle, of Carnegie Institute of Technology, and from A. C. Fieldner and W. P. Yant, of the U. S. Bureau of Mines, during the preparation of this paper. The following tables give a general classification of the gas-detecting appliances now on the market:

FOR DETECTING METHANE

APPLIANCE	LOWEST PERCENTAGE DETECTABLE	FIELD OF USE	REMARKS
Flame Safety lamp	1.0-2.0	General inspection, fire-boss, etc.	Roughly quantitative, depending on lamp, fuel and operator.
Burrell indicator	0.1	Foreman, asst. foreman, development entries, return air of splits, etc.	Accurate in trained hands.
Gas-analysis apparatus	0.02	Gas inspector, chemist, return air of splits, etc.	Accuracy depends on laboratory technique of operator.
Continuous recorder	0.01	Fan house, etc.	Accurate, automatic.

FOR DETECTING CARBON MONOXIDE

Canary bird	0.1-0.15	General recovery work, etc.	Roughly quantitative, depending on bird and length of exposure.
Carbon monoxide detector	0.07	General recovery work, fire fighting.	Fairly accurate, depending on ability to read color scale.
Pyrotannic detector	0.01	Safety engineer-fire fighting, etc.	Accurate in trained hands.
Gas-analysis apparatus	0.01	Safety engineer or chemist; fire fighting, etc.	Accuracy depends on laboratory technique of operator.
Continuous recorder	0.0004	Fan house spontaneous oxidation, etc.	Accurate, automatic.

DISCUSSION

D. C. ASHMEAD,¹ Wilkes-Barre, Pa. (written discussion).—Two instruments designed for the detection of methane in coal mines have been added to the list of mine safety devices. The first was designed to be operated by chemical reaction and was the outcome of research work by E. K. Judd, of the Union Carbide and Carbon Research Laboratories, Inc. This instrument was found accurate and entirely satisfactory except that its action was not quite as speedy as was desired. A second

¹ Mr. Ashmead's discussion was followed by a demonstration of the detector and recorder.

instrument was then developed by A. N. Erickson, also of the Carbide laboratories, to be operated electrically, which reacted instantaneously and accurately under actual testing conditions.

The electrical methane detector was the instrument demonstrated before the ventilation session of the Institute held in New York City on Feb. 15, 1927. A description of the operation of this instrument was given in the March issue of *MINING AND METALLURGY*, and the construction was shown by the cut reproduced here.

The methane detector consists of a four-volt storage battery, a resistance box and a combustion chamber. The resistance box contains three resistances, *B*, *C* and *E*, and the filament in the combustion chamber acting as the fourth resistance forms a Wheatstone bridge. A combination switch, *HI*, is used, which automatically turns on the current to the filament before throwing the meter in the circuit. The meter, *G*, is calibrated directly in per cent. methane. The battery is connected to the resistance box by means of a small, portable, thoroughly insulated jack and plug. This is not shown on the diagram. This method of connection is used so that the battery may be disconnected easily from the resistance box.

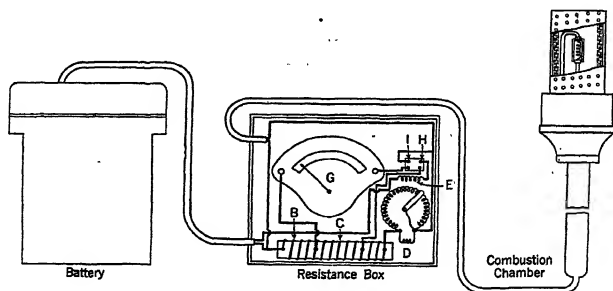


FIG. 6.—ELECTRICAL METHANE DETECTOR.

The filament, of platinum wire, is arranged in the form of a holder, which may be replaced easily. The combustion chamber consists of a bonnet, three concentric screws spaced evenly, and a base to which the filament holder and the leads from the resistance box are connected. Attached to the base is a telescopic handle, which makes it possible to lift the detector head of the apparatus to the roof or other high places.

When the combustion chamber is brought into an atmosphere containing methane, the gas burns on the filament, raising its temperature in proportion to the percentage of gas present in the atmosphere. The increased heat of the filament causes a resistance to the flow of electric current in proportion to the heat generated by the burning gas, and this results in a flow of some current through the meter. The greater the increase of resistance, the greater the flow of current through the meter. The flow of current through the meter is registered on a dial which is calibrated to read in percentage of methane. The apparatus is light and compact. Its reaction in the presence of methane is almost instantaneous and in actual practice amounts as small as 0.16 per cent. have been readily detected.

F. HAAS, Fairmont, W. Va.—I am impressed with the accuracy of the machine described by Mr. Ryan, which he states is within 0.01 per cent.—everything that could be demanded in an operating way. I have looked over the description of the machine and followed its operation carefully and as far as I can go I find the theory correct and the machine practical. It lacks, however, the portability feature and therefore does not seem to have practical application. The total amount of methane in a mine

is not of so much interest as the percentage of gas that is given off on each individual split. If a mine has considerable gas, that gas must be controlled. The machine in itself does not give us that control, though a statement to that effect was made, and I do not understand how it can give control. Moreover, the machine is comparatively expensive to start with and would require expert attention to keep it in working order.

Comparing it with the machine described by Mr. Judd, we find that the latter machine is portable, inexpensive but lacks sensibility. The combination of the accuracy of the one machine with the recording and portability features of the other would solve the entire problem I have been working on for 20 years at least.

D. C. ASHMEAD.—It is hoped that the machine just demonstrated will weigh only about 7 lb. complete when it is put on the market.

D. HARRINGTON, Washington, D. C.—I was particularly interested in the demonstration by Mr. Ashmead. The people who perfect a methane detector will certainly be benefactors to the entire mining industry. What will be the cost of that machine?

D. C. ASHMEAD.—Probably less than \$100.

D. HARRINGTON.—Does the 7 lb. include the battery?

D. C. ASHMEAD.—Yes, the machine complete.

D. HARRINGTON.—Mr. Steidle, can you give the cost of the continuous recording devices, both as to carbon monoxide and methane?

E. STEIDLE, Pittsburgh, Pa.—Size, requirements, portability, etc., can be taken care of fairly well. The cost at present of the continuous methane recorder is about \$600, and the carbon monoxide recorder about \$1200.

D. HARRINGTON.—Mr. Ashmead, have you placed this machine before the Bureau?

D. C. ASHMEAD.—No. We hope to do so in the next 60 days.

T. D. THOMAS, Lansford, Pa.—In the anthracite regions there are pitches of from 40° to 60°. Only portable machines can be taken into such places.

It would certainly be desirable to have a machine for detecting methane in splits. Control of the amount of air, if there is an excess amount of methane, is the most desirable feature. If the fan is electrically driven with a fixed pulley there will be difficulty in speeding it up. All of these machines tell very nicely the content of the air, but to get a fan automatically controlled with the detecting machine would be very beneficial to the mining industry, for many reasons.

T. G. FEAR, Fairmont, W. Va.—I am very much interested in the development of both of these machines because usually we all take our samples at a predetermined time. A good many mines take them at the same time every day, usually before the men enter the mine. The men who have traveled around the mines both day and night know that between 3 and 7 o'clock in the morning the mine is quiet, as cutting is usually done between four in the afternoon and 3 o'clock in the morning. There is less chance of gas being given off in new working places between 3 and 7 o'clock in the morning than between 7 and 10 o'clock, when the shooting is usually done.

I operated Mr. Ryan's machine for several months and found it checked accurately with the samples we took and had analyzed in our laboratory. They checked within 0.01 per cent. with the laboratory samples, but as the methane content of the main return was only around 0.18 per cent., I could not find any difference in the content during the time of the shooting and when the mine was quiet.

W. P. YANT, Pittsburgh, Pa.—The two recorders in question operate on entirely different principles. For example, the carbon monoxide recorder functions by measuring the heat of reaction produced by oxidizing carbon monoxide to carbon dioxide. Oxidation is effected by passing the gas through a capsule or cell filled with a suitable catalyst, in which are embedded a number of thermocouples. Changes in temperature are recorded by a potentiometer, the scale of which is calibrated to read in parts CO rather than in millivolts. The sensitivity may be increased or decreased depending on the number of thermocouples and the type of potentiometer used.

On the other hand, the methane recorder, as I understand it, operates on the principle of the change in volume produced in a methane-air mixture by burning the methane. The sensitivity of this apparatus, therefore, appears to be limited by fixed factors, as the volume change for a given amount of methane and the refinement of flow meters for determining the volume change. To obtain checks to 0.01 in the range of 0.18 per cent. methane is indeed remarkable. It is much better than we ordinarily obtain in the use of devices of this type for gas measurement.

J. T. RYAN (written discussion).—Referring to Mr. Haas' very constructive comments on the methane recorder described in my paper: The original thought back of the development of the methane recorder was to make it available for use underground on the separate splits as well as for use on the main return. The installation of the machine underground would necessarily require that it be entirely flameproof, which would require a flameproof motor and electric furnace. There was no flameproof motor available in the small size required and no flameproof furnace available and for that reason we have only advocated installing the machine on the surface.

Further research work since the time the paper was prepared has proved that we can draw a sufficiently large sample to operate the machine through 5000 ft. of $\frac{1}{4}$ -in. brass tubing and with comparatively little lag in the machine as the pump which draws in the sample will operate under $29\frac{1}{2}$ in. of suction. It is possible, therefore, to have the machine described in the paper located on the surface and draw the samples from a distance of 5000 ft., and even a greater distance by using a larger size tube.

Referring to Mr. Yant's comments on the sensitivity of the machine which, as he states, is largely a factor of the refinement of the flow meters. The maximum error allowed in the flow meters is 5 per cent. With a mixture containing 0.2 per cent. methane this maximum error of 5 per cent. would only make an error of 0.01 per cent. in the methane determination from this cause. Experience gained from a number of installations that have been checked indicates that the average sensitivity is about 0.01 per cent., but should all the errors from these fixed factors as mentioned by Mr. Yant be in the one direction or accumulative they would amount to 0.03 per cent., which we now state is the accuracy of the machine.

Factors in the Ignition of Methane and Coal Dust by Explosives*

By G. ST. J. PERROTT,† PITTSBURGH, PA.

(Pittsburgh Meeting, October, 1926)

ONE of the important hazards in coal mining is the danger of ignition of explosive mixtures of methane and air or coal dust and air, or both, by the explosives used in blasting the coal. It has long been recognized that certain explosives are more dangerous than others in this respect. In America and abroad, government testing stations have been established for officially passing on the suitability of explosives for use in coal mines. In this country the Federal government is given no power to enforce the use of the approved explosives but merely publishes from time to time, through the Bureau of Mines, those explosives which have passed the tests necessary to place them on the list for use in coal mines. In spite of this lack of police power, the increase in the use of permissible explosives since the promulgation of the first list in 1909 has been rapid. In 1912, only 8 per cent. of the explosives used in coal mines were permissible; in 1925, this figure had increased to 25 per cent.

While the past record of permissible explosives has been most gratifying, it is very desirable that we pave the way for further improvement in explosives and blasting methods by learning more about the mechanism of ignition of gas and coal dust by explosives. To this end a considerable amount of investigative work already has been carried out and a large amount yet remains to be done.

It is the purpose of this paper to discuss briefly the effect of the characteristics of explosives and method of loading in the borehole on the liability of a blown-out shot to ignite explosive mixtures of gas or dust. For an account of the methods by which an explosive is tested for permissibility for use in coal mines, the reader is referred to *Schedule 17-A* of the U. S. Bureau of Mines.

SAFETY FACTOR IN USING EXPLOSIVES

The factors that affect the liability of an explosive to ignite gas or dust are already known in a qualitative way but quantitative data are

* Published with approval of the Director, U. S. Bureau of Mines.

† Assistant chief explosives chemist, U. S. Bureau of Mines, Pittsburgh Experiment Station.

for the most part lacking. It is generally conceded that the relative safety of an explosive as regards igniting combustible mixtures depends upon (1) temperature, size and duration of flame, and (2) heating of the combustible mixture, produced by adiabatic compression, as the hot gases are rapidly discharged from the borehole. These factors not only depend upon the character of the explosive but also upon the position of the borehole with regard to the entry, the size and form of entry or room in the mine, the method of loading in the borehole, its diameter and length, size of cartridge, and type of stemming employed. A permissible explosive differs from dynamites and from black blasting powder in that it produces a smaller flame of lower temperature and shorter duration. This type of explosive has often been called "flameless," but as a matter of fact all explosives when fired as a blown-out shot produce a visible flame which can be photographed.

The greater part of the permissible explosives used in America are of the ammonium nitrate-nitroglycerin type, and contain about 10 per cent. each of nitroglycerin and wood pulp, and the remainder ammonium nitrate, plus from 2.5 to 15 per cent. of sodium nitrate or sodium chloride. The latter ingredients, particularly sodium chloride, are added to reduce the size and temperature of the flame. The rates of detonation of the explosives range from about 1800 to 4000 m. per sec. The change in rate from low to high is usually brought about by employing ammonium nitrate of different physical characteristics.

FLAMES OF VARIOUS EXPLOSIVES

Fig. 1 shows the difference between the flames produced by permissible explosives and other explosives. These are photographs of flames produced by charges of explosives fired from a steel cannon into the air. In the shot of black powder, 1 lb. of stemming was used whereas in the case of the other two explosives, $\frac{1}{4}$ lb. stemming was used. A larger quantity of stemming cools the flame from detonating explosives to such an extent that photography is difficult. The weights of permissible explosive and 40 per cent. nitroglycerin dynamite used were equal in strength as determined by the ballistic pendulum,¹ whereas the quantity of black powder used was about one-fourth the strength of the charge of the other two explosives. A larger quantity of black powder gave a flame too voluminous for the scale of the photograph, but even from this amount of black powder it will be seen that the flame is considerably larger than for the other two explosives and evidently extremely likely to ignite explosive mixtures of air with gas or coal dust. The flame from the

¹ *Bull.* 15, Bur. Mines (1912) 79. ("Unit deflective charge" is defined as weight of explosive which gives same swing of pendulum as 227 g. ($\frac{1}{2}$ lb.) of Bureau of Mines' 40 per cent. standard dynamite.)

permissible explosive is considerably smaller than that from the 40 per cent. dynamite. Photographs taken on a moving film show further that the flame of the shot of black powder is of considerably longer duration than that of the dynamite and permissible explosive.

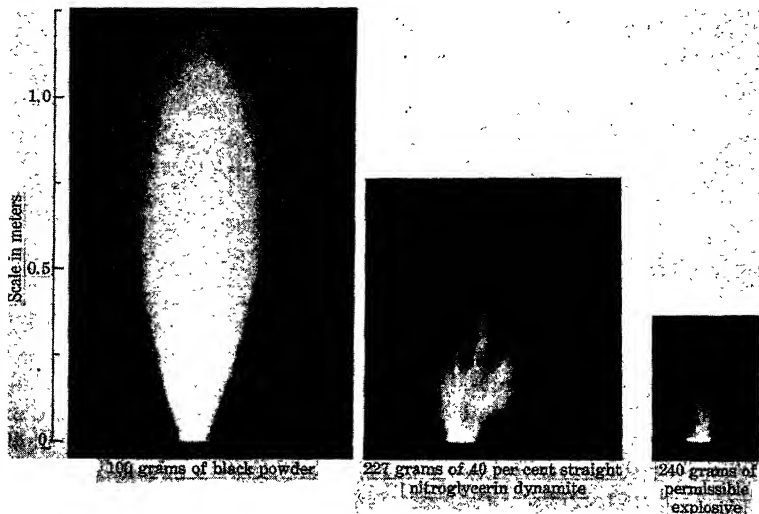


FIG. 1.—FLAME PHOTOGRAPHS OF STEMMED SHOTS OF EXPLOSIVES FIRED FROM A CANNON.

METHOD OF TESTING EXPLOSIVES

The flame photographs give a qualitative indication of the safety of the explosives as determined at the Explosives Testing Station at the Bureau's Experimental Mine, in the gas and dust gallery. This is a cylindrical steel chamber 6 ft., 4 in. in diameter and 100 ft. long. One division, 20 ft. long, may be separated from the remainder by a diaphragm, thus forming a chamber which contains the explosive mixture of gas and air or coal dust and air, or both. A charge of explosive is fired from the borehole of a steel cannon into this division of the gallery to determine whether or not the explosive will ignite the mixture. The borehole is $2\frac{1}{4}$ in. in diameter by $21\frac{1}{2}$ in. long, and has a capacity of as much as 2 lb. of explosive with space for stemming if desired.

DETERMINATION OF LIMIT CHARGE

In determining the relative safety of different explosives by this test or for a given explosive to find the effect of different methods of loading or of various concentrations of gas and air, what is known as the "limit charge is determined." This is defined as the maximum weight of explo-

sive that will not ignite the mixture in the gallery in five or more shots. Greater weights than the limit charge do not always cause ignition until the weight used is considerably greater than the limit charge.

In obtaining the data in Table 1, the first 20-ft. division of the gallery contained a mixture of 8 per cent. natural gas² and air. The natural gas was of approximately the following percentage composition: Methane, 88; ethane, 7; propane, 2.5; butane, 1, and nitrogen, 1.5. Explosive A, of the following description, was employed: Chemical composition—nitroglycerin, 10; ammonium nitrate, 79; calcium carbonate, 1, and wood pulp, 10 per cent.; grams of wrapper per 100 gm. of explosive, 8.0; density, 0.97 g./c. c.; rate of detonation, 3500 m. per sec. Unless otherwise stated, 1 lb. of stemming was used.

TABLE 1.—*Limit Charges in 8 Per Cent. Natural Gas; Effect of Method of Loading and Stemming*

Stemming	Method of Loading	Limit Charge, Grams
None.....	Density-of-one ^a	20
Dry fireclay.....	Density-of-one ^a	225
Moist fireclay ^b	Density-of-one ^a	400
Dry fireclay.....	8-in. air space	125
Dry fireclay.....	4-in. air space	175
Dry fireclay.....	Cartridge laid in borehole	175
Moist fireclay ^b	8-in. air space	375
Rock dust.....	Density-of-one ^a	500
Coal dust (½ lb.).....	Density-of-one ^a	150

^a Density-of-one indicates explosive was tamped tight into borehole.

^b 1.4 oz. water per lb. of stemming.

The increased safety of stemmed shots over unstemmed shots is shown by a limit charge of 20 gm. for the unstemmed shot as against 225 gm. for a shot tamped with 1 lb. of dry fireclay stemming. Fine rock-dust stemming is apparently more efficient than the fireclay, as evidenced by a limit charge of 500 gm. A half-pound of coal-dust stemming gives a limit charge of 150 gm., but coal-dust stemming is out of the question on account of its great hazard. A little moisture added to the fireclay stemming increases the limit charge markedly. An 8-in. air space between the explosive and fireclay stemming reduces the limit charge to 125 gm. The same effect is evident, but to a less extent, when the cartridges (1¼ in.) are laid in the borehole (2¼ in.) but not compressed to fill the hole. The limit charge under these conditions is 175 gm., as against 225 gm. for a similar shot with no air space between explosive and borehole.

² Calculated on nitrogen-free basis and figured as methane plus ethane from data of an analysis by slow combustion pipette in Orsat apparatus.

The loading density for the density of one shot is about 0.9 gm. per c. c.; with the 8-in. air space it is 0.13 gm. per c. c.; when cartridges are laid in the borehole, it is 0.23 gm. per c. c.

It should be borne in mind that the gallery test is carried out under standardized conditions and that other factors may affect the results obtained under practical mining conditions. For example, the shot is fired from an unyielding steel borehole, hence it is always a blown-out shot whatever the nature of the stemming material. In blasting coal, other things being equal, a blown-out shot is least likely to occur with the stemming material which offers best confinement. This factor can not be evaluated in the gallery tests.

ANGER OF COAL-DUST STEMMING

The danger of using coal-dust stemming³ has been well demonstrated in a recent series of tests⁴ in which 1½ lb. of a permissible explosive was fired into the gallery which contained 4 per cent. gas and 20 lb. of coal dust on the shelves. With no stemming, this explosive caused no ignitions in 10 shots. With 1 lb. of coal-dust stemming, two ignitions were obtained out of 13 shots; with ½ lb., six ignitions out of 10 shots; and with ¼ lb. five ignitions out of six shots. The first condition with no stemming is identical with the conditions of Test 4, which is one of the tests all explosives on the permissible list must pass. It is evident from the above results that coal-dust stemming increases the danger over that existing when no stemming is used. While the larger amounts of coal-dust stemming do not give rise to ignitions as frequently as do smaller amounts, it is evident that the use of combustible stemming always presents an increased hazard.

RELATIVE SAFETY OF METHODS OF LOADING

Fig. 2 illustrates the relative safety of the several methods of loading. In all cases 330 gm. of the explosive was used and 1 in. of the borehole filled with stemming. This was about 150 gm. of the incombustible stemming and 80 gm. of the coal dust. More stemming was not used because the flame is thereby cooled to a reddish color which does not photograph.

It is interesting to speculate on the reason for the decreased limit charge when an air space exists between explosive and stemming. Tests in a specially designed apparatus,⁵ by means of which the gases produced when a charge of explosive is fired from the cannon can be caught for

³ Coal dust used in all tests was 98 per cent. through 100 mesh and 85 per cent. through 200 mesh.

⁴ J. E. Tiffany: Bur. Mines unpublished report.

⁵ J. E. Crawshaw and G. W. Jones: An Apparatus for Studying Gases of Explosives Under Confinement. *Eng. & Min. Jnl.-Press* (1925) 120, 965.

analysis, have shown that considerably more combustible gases are formed by an air-spaced shot than by a density-of-one shot. This is undoubtedly due to the lower pressure in the former case which tends toward the production of relatively more carbon monoxide and hydrogen. The higher percentage of inflammable gases produced without any great lowering of

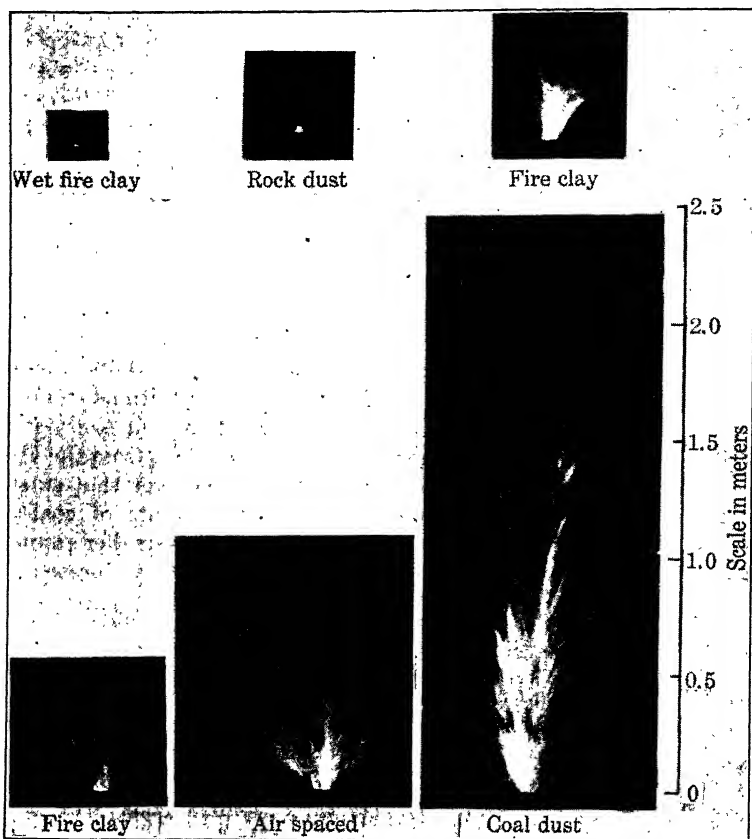


FIG. 2.—FLAME PHOTOGRAPHS SHOWING EFFECT OF CHANGE OF STEMMING AND METHOD OF LOADING.

flame temperature may be responsible for the lower limit charge with air-spaced shots, or, on the other hand, it may be due to some purely mechanical cause, for example, the manner in which the stemming is projected from the borehole in the two cases. Table 2 gives the gaseous products of detonation produced by Explosive A under the two methods of loading.

TABLE 2.—*Gaseous Products of Detonation from Two Methods of Loading*

Method of Loading	Weight of Explosive	Liters of Gas at 0° and 760 mm. Hg from 100 Gm. Explosive				Carbon Deposited, Gm. per 100 Gm. Explosive
		CO ₂	CO	H ₂	CH ₄	
Density 1.....	156.6	11.5	4.6	3.9	3.7	1.8
8-in. air space.....	159.6	11.5	9.6	9.2	2.0	0.0

EFFECT OF CHARACTER OF EXPLOSIVE ON LIMIT CHARGE

As little as 5 gm. of black powder tamped with 1 lb. of dry fireclay stemming will ignite a mixture of 8 per cent. natural gas and air in the gallery. Under similar conditions, 150 gm. of 40 per cent. straight dynamite is required; all permissible explosives require a larger quantity than the unit defluctive charge. The limit charge of the ammonium nitrate explosives is raised by the addition of a cooling agent such as sodium chloride or by so changing the composition that there is an oxygen deficiency which produces more carbon monoxide when the explosive is detonated, thereby giving a flame of lower temperature.

For explosives of identical chemical composition, those with the higher rate of detonation have the lower limit charge. For example, two explosives of the ammonium nitrate-nitroglycerin class, containing 10 per cent. salt, detonated at rates of 2100 and 3300 m. per sec., respectively, and showed limit charges in 8 per cent. natural gas of 350 gm. and 275 gm. In another group of similar explosives without salt, the rates varied from 2200 to 4000 m. per sec. and the limit charges from 250 to 175 gm., respectively.

It is more difficult to determine the limit charges in mixtures of coal dust and air in the absence of gas or in the presence of low percentages of gas, because of the large quantities of explosives (of the permissible type) which must be used. Experiments in which the gallery contains 4 per cent. gas and 20 lb. of coal dust on the shelves, show that 25 gm. of black powder will cause an ignition, whereas 680 gm. (1½ lb.) of a permissible will not cause an ignition.

EFFECT OF GAS CONCENTRATION ON LIMIT CHARGE

Fig. 3 shows the relation between limit charge and concentration of natural gas for Explosive A previously mentioned. When the explosive is fired in its original wrapper, the most sensitive mixture is between 7½ and 8 per cent. gas and the curve is fairly flat between 7 and 8½ per cent. gas; in other words, variation of gas concentration between these limits does not affect the limit charge to a large extent. Change in concentration beyond these limits rapidly decreases the sensitivity of the gas mixture, but the change is less rapid for concentrations lower than 7

per cent. than is the change as the concentration is increased above $8\frac{1}{2}$ per cent. This is probably due to the fact that the combustible gases given off by the explosive tend to raise the concentration of the gas mixture, the effect of which is to increase the sensitivity of gas mixtures of concentrations lower than the most sensitive mixture.

This point is well corroborated by results obtained when this explosive is fired without its wrapper. The explosive now contains slightly more oxygen than necessary for complete oxidation of the carbon and hydrogen to carbon dioxide and water. The effect of this is to reduce the limit charge at gas concentrations between 7 and 9 per cent. and to raise it

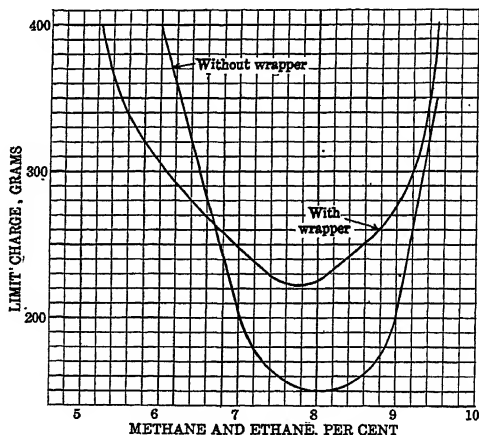


FIG. 3.—RELATION BETWEEN CHARGE LIMIT AND GAS CONCENTRATION FOR A PERMISSIBLE EXPLOSIVE.

at concentrations of 6.5 per cent. and lower, and to make the most sensitive mixture a somewhat higher concentration than in the case of the explosive with wrapper. The reduction of the limit charge at the concentrations around 8 per cent. is due to the fact that the flame of the explosive fired without wrapper is hotter (calculated temperature, 2670° C.) than the flame produced when fired with wrapper (calculated temperature, 2070° C.), whereas the apparently paradoxical behavior at concentrations below 6.5 per cent. is due to the fact that in the case of the explosive without wrapper, no combustible gases are produced to increase the gas concentration in the gallery while such gases are produced by detonation of the explosive and wrapper. Thus, the effective concentration in the latter case is higher than that present in the gallery before the explosive charge is fired. The curve in the former case is thus fairly symmetrical around the most sensitive concentration, while in the latter case it is not. We should expect an explosive that contains a large

excess of oxygen to give a steep curve on the low side of the most sensitive mixture and a less steep curve on the high side; in other words, the reverse of that obtained with the present explosive when fired with wrapper.

It is obviously desirable that we know more about the causes that effect ignition of gas and coal dust by explosives, and the factors that affect the liability of ignition. As previously pointed out, these are already known in a qualitative way but the relative importance of the various factors is unknown, and measurement of temperatures and pressures produced has not been consummated as yet.

FLAME TEMPERATURES

Temperature may be calculated by making use of a number of assumptions, but it is questionable how close calculated values check actual temperatures, in view of the lack of information of the conditions prevailing in the flame. A direct method of measuring flame temperature is much needed. Calculated temperatures of the gaseous products of detonated permissible explosives range from 2100° C. to 2300° C., whereas 40 per cent. straight nitroglycerin dynamite has a calculated temperature of 2600° C.

The duration of the flame may be measured by recording the image of the flame on a rapidly moving photographic film. Fig. 4 shows such photographs. In taking the photographs, a vertical slit about $\frac{1}{16}$ in. in width is placed in front of the rotating film. The duration of the flame, as calculated by Bichel,⁶ was measured horizontally from the mid-point of the bottom of the image to the projection of the top of the image; in other words, it was the total time that any flame was in existence.

PRESSURE DEVELOPED BY EXPLOSIVES

There is no method at present for measuring pressures produced near the mouth of the cannon. Dixon⁷ has shown, however, that a pressure of 54 atmospheres suddenly applied is sufficient to ignite a mixture of 6.5 per cent. methane and air. It is obvious that a sufficiently large charge of an explosive which produced no flame whatever might under these circumstances ignite an explosive mixture by compression alone.

The relatively greater sensitiveness of gas mixtures to ignition in galleries of smaller diameter is due to the increased compression and consequent greater heating of the combustible mixture in the small gallery. It is also evident that, other factors being equal, an explosive that has a high rate of detonation might be expected to ignite a gas and air mixture more readily than one having a lower rate, because the compression (of

⁶ C. E. Bichel: *New Methods of Testing Explosives*. (English Translation.) Charles Griffin & Co. (1905) 42.

⁷ Second Report Explosives in Mines Research Committee, Safety in Mines Research Board, London (1925) 6.

the combustible mixture) at the mouth of the borehole would be greater in the former case. This is borne out by tests in the gallery. However, when the rate of explosion becomes as low as in the case of black powder, the factor of flame duration has a greater effect and the explosive becomes more dangerous.

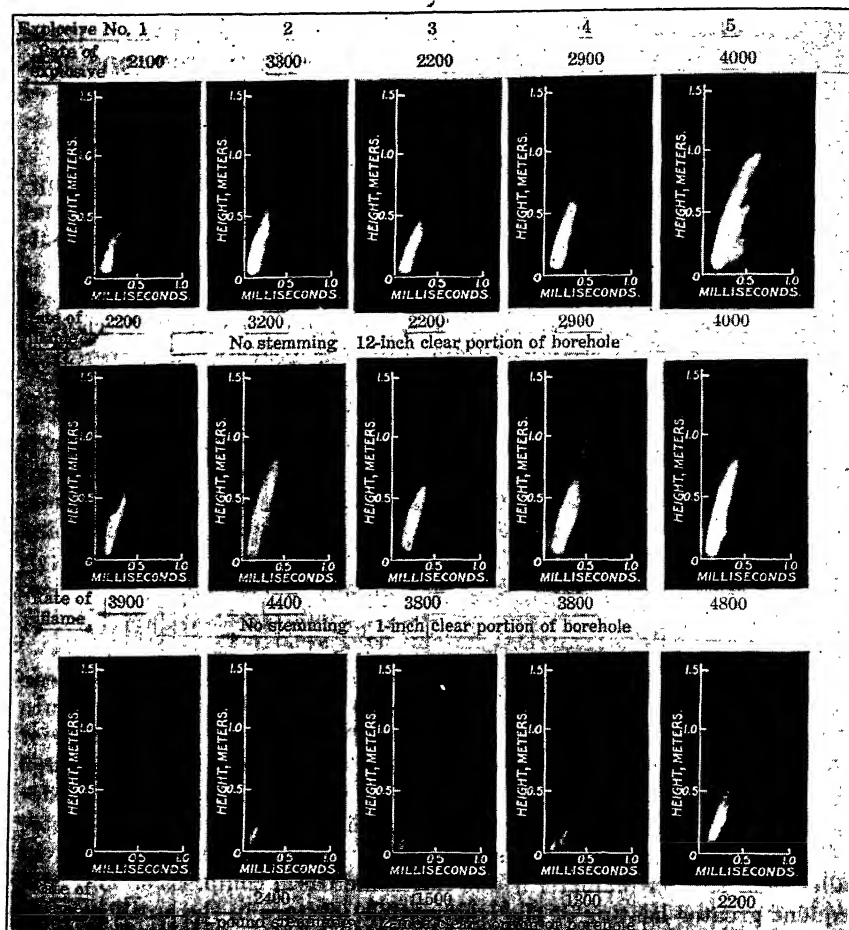


FIG. 4.—FLAME PHOTOGRAPHS OF PERMISSIBLE TYPE EXPLOSIVES TAKEN ON A MOVING FILM.

In this connection, photographs on a moving film of the flames of explosives fired from a cannon are of interest. Fig. 4 shows pictures for five ammonium nitrate explosives. Explosive 1 has a limit charge in 8 per cent. natural gas-air mixture around 350 gm.; explosive 2. 275 gm.:

explosive 3, 250 gm.; explosive 4, 225 gm.; and explosive 5, 175 gm. A unit defective charge was employed, or about 240 gm. for explosives 1 and 2, and 215 gm. for explosives 3, 4 and 5. The first horizontal line of pictures shows the result when the explosive was at the bottom of the borehole. If the rate of projection of the flame for the first 5 cm. is measured, it is found that this rate is about the same as the rate of

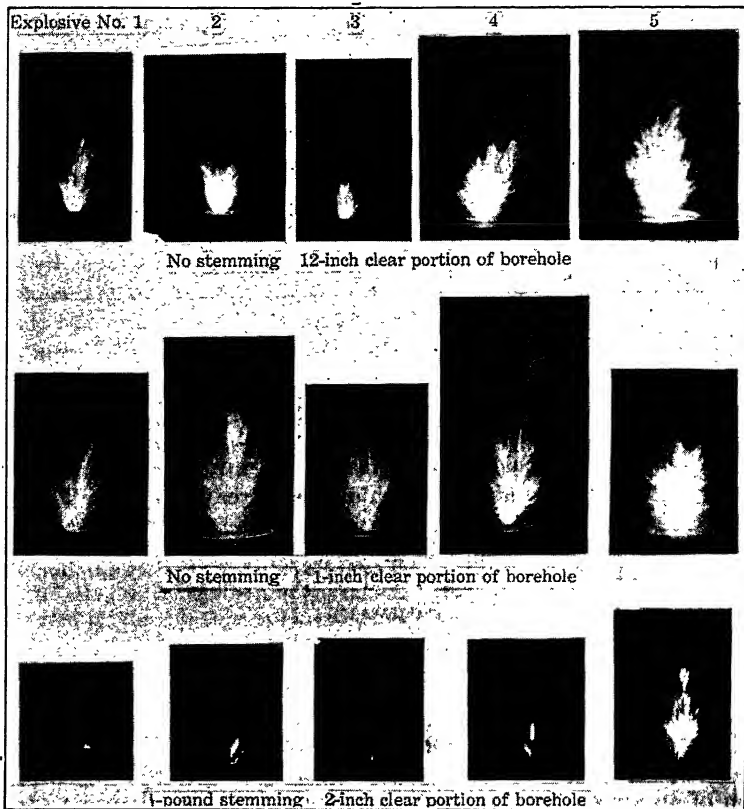


FIG. 5.—FLAME PHOTOGRAPHS OF PERMISSIBLE TYPE EXPLOSIVES TAKEN ON A STILL PLATE.

detonation of the explosive. On the other hand, when the borehole is shortened by filling up with fireclay, as in the second line of pictures, the rate of projection of the flame is considerably faster than the rate of detonation of the explosive. With 1 in. of stemming, the rate is considerably slower than that of the explosive. The photographs in Fig. 5 show the same shots but taken with an ordinary fixed plate camera.

VALUE OF PHOTOGRAPHY IN EXPLOSIVES TESTING

For phenomena that take place in such short periods, photographic methods seem particularly promising. Photography of flames from explosives fired into air has given results which serve to divide explosives into groups as regards safety, but does not always distinguish between members of the groups. It is planned to take photographs on a moving film of the ignition of gas-air and coal dust-air mixtures by explosives in a specially constructed gallery, 6 ft., 4 in. in diameter and 20 ft. long, with a horizontal slot along the center closed by plate glass windows. By means of this apparatus it is hoped to obtain further knowledge of the mechanism of ignition of gas and coal dust by explosives.

The tests established in 1908 have resulted in the development of permissible explosives which have had a remarkable record in actual practice both from the standpoint of efficiency and of safety. However, it is believed that a more complete knowledge of the mechanism of detonation of explosives and of the mechanism of ignition of gas and dust by explosives should result in the development of explosives and methods of using them having a yet larger factor of safety. It is to this end that the Bureau of Mines is investigating certain fundamental properties of explosives and of the ignition of gas and coal dust, in addition to carrying out the present schedule of official tests. Invaluable work along similar lines is being carried out in Great Britain, France and Belgium. The difficulties are great but the goal of greater safety in mining is well worth winning, and is the common aim of much of the Bureau's work. This paper has indicated some of the lines along which the explosives investigations in this direction are being conducted.

SUMMARY

While the investigation, of which this discussion is largely a progress report, is far from being complete, yet the results obtained warrant the following conclusions which apply to blown-out shots:

1. That the method of loading and the kind and condition of stemming have an important effect on the relative safety as follows: (a) The introduction of an air space between the explosive and the stemming reduces the safety to a slight degree; (b) the conditions which furnish the greatest relative safety are loading the explosive tight in the borehole and the use either of a distinctly moist, inert material, such as damp fireclay or a finely pulverized inert material such as rock dust; (c) the use of coal dust as stemming increases the danger of ignition of gas or dust by a blown-out shot.

2. An explosive gas mixture containing $7\frac{1}{2}$ to 8 per cent. of natural gas is the mixture most sensitive to ignition by the permissible explosives commonly used in coal mines in this country, but between 7 and $8\frac{1}{2}$ per

cent. there is little practical difference in the sensitiveness of the mixture. On either side of these limits the sensitiveness decreases rapidly. With those gas mixtures most sensitive to ignition (7 to $8\frac{1}{2}$ per cent. of gas), a balanced explosive is most likely to cause ignition but an explosive which is under-oxidized is more likely to cause ignition of gas mixtures near the lower limit of inflammability, which is the condition commonly met with in practice.

3. Definite indications have been obtained that rate of detonation is an important factor in the ignition mechanism and that the explosive having the higher rate of detonation may be expected to ignite a gas and air mixture more readily than one having a lower rate of detonation.

4. Photography of flames produced by explosives fired into air gives results which serve to divide the explosives into groups as regards safety, and taken in connection with the composition of the explosive and the rate of detonation promises to throw light on the mechanism of ignition.

DISCUSSION

F. HAAS, Fairmont, W. Va.—Under what conditions will 20 atmospheres of pressure ignite an explosive mixture?

G. ST. J. PERROTT.—Under conditions of adiabatic compression, Professor Dixon in England found that a pressure of 54 atmospheres suddenly applied would ignite a mixture of 6.5 per cent. methane and air while as little as 28 atmospheres pressure was sufficient to ignite a mixture containing 7.5 per cent. methane and 92.5 per cent. air. The experiments were made in a steel cylinder and the pressure applied suddenly by dropping a weight onto a piston fitted in the cylinder. The English believe that many of their disasters in coal mines in Great Britain may have been due to some such cause. Very often in coal mining practice in Great Britain, there will be a fissure or fault containing gas near the borehole and many of the disasters in Great Britain have been attributed to the ignition of gas in this pocket at first, which possibly may have been caused by the pressure produced when the shot was fired.

F. HAAS.—You have answered my question; if I get it correctly, that this pressure must be applied rapidly.

G. ST. J. PERROTT.—Very rapidly, yes.

F. HAAS.—If the pressure were to be applied slowly and maintained, there would be no ignition.

G. ST. J. PERROTT.—No. For an isothermal compression, such an enormous pressure would be required that it probably would be impossible to attain.

T. L. MCCALL, Stellarton, Nova Scotia.—Mr. Perrott showed how the danger was increased by air cushioning. I would like to ask him to what extent this air cushioning was in the cannon.

G. ST. J. PERROTT.—The air cushioning was entirely in the cannon. The explosive was loaded first into the bore hole of the cannon, then an 8-in. air space was introduced by means of a wire spacer, and then the stemming was put in.

MEMBER.—That would be equivalent then to using say a $1\frac{1}{4}$ -in. diameter charge, $1\frac{1}{2}$ -in. hole?

G. ST. J. PERROTT.—Yes, we have tried that and the effect is in the same general direction.

T. L. McCALL.—Cushion blasting is of great interest to operators who are trying to get a larger quality of lump coal and you put rather a damper on us when you tell us it is more dangerous.

G. ST. J. PERROTT.—We do not have sufficient data as yet to permit applying our results to practice. Our results apply only to this particular set of experimental conditions. In the gallery test, we always have a "blow-out" shot, while in practice the blow-out shot is the exception. I should say that a cushion shot which did not blow out would be just as safe as a non-cushion shot.

T. L. McCALL.—Then there is another question I would like to ask. Take a permitted explosive. Say that the permitted charge is 34 oz. but the operators only use 16 oz. as their limit, and then they decide to go to 20 oz. Are you increasing your danger by going to 20 oz. from 16 oz.?

G. ST. J. PERROTT.—Yes, if your conditions of loading and the amount of burden on the hole remain the same.

T. L. McCALL.—But if the burden on the hole is increased?

G. ST. J. PERROTT.—Then you may not be increasing your danger. Obviously, the danger depends on the extent to which you approach a blow-out shot. If you have sufficient burden on your hole for the gases to be cooled before they are liberated, then you can fire quite a large quantity of explosive and still be safe. But other things being equal, danger of ignition increases as the weight of charge is increased.

G. ST. J. PERROTT (written discussion).—Further experimental work carried on since this paper was written has shown that the relative safety of a cushion shot depends greatly on the method by which stemming and air space are introduced and that no generalization can be made to the effect that cushion blasting is more dangerous than non-cushion blasting. These data will be included in a future publication.

Use and Dangers of Booster and Auxiliary Fans as Applied to Coal Mine Ventilation*

By H. I. SMITH,† WASHINGTON, D. C.

(New York Meeting, February, 1927)

THE technical and safety press have devoted much space in support of or in opposition to the use of booster and auxiliary fans in coal mines. The Mine Safety Board of the U. S. Bureau of Mines has given the matter consideration, and the Mine Ventilation Committee of the A. I. M. E. is now seeking a discussion of the merits and demerits of such fans to guide it in formulating recommendations. In this paper I will set up some principles on which I believe most engineers will agree and from which we may work toward a decision, though as to what that decision shall be there may be a difference in opinion. Before stating the points on which I believe there is accord, I will distinguish between booster and auxiliary fans.

DEFINITION OF BOOSTER AND AUXILIARY FANS

In discussing underground fans G. S. Rice¹ distinguishes between those used to increase the amount of air passing a given point and those



FIG. 1.—A BOOSTER FAN SHOWING BY-PASS DOOR OPENED SO THAT MOTOR AND BELT CONNECTION CAN BE SEEN.

used to augment the ventilation locally. He describes a booster fan as one which is installed to handle the entire air current passing a given

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† Chief mining supervisor, U. S. Geological Survey.

¹ Chief mining engineer, U. S. Bureau of Mines.

point in the air course and which is intended to increase the volume of air that passes that point. One type of booster fan is shown in Fig. 1.

An auxiliary fan is a fan installed to divert a part of the air current passing a given point for the purpose of ventilating some particular place or places, while allowing the remainder of the air current to pass by.

POINTS GENERALLY ACCEPTED AS FUNDAMENTALS

The fundamentals of mine ventilation on which I believe there is general agreement are as follows:

1. Booster fans will recirculate air if there are any leaky stoppings, overcasts or doors between the intake and return airways on the suction side of the fan or if there is loose construction in the fan housing.

2. Where air is recirculated there is an undesirable decrease in the percentage of oxygen and an increase in the percentage of methane in the ventilating current. The former reduces the amount of work which men and animals can perform, and the latter is a hazard to the lives of the men in the mine.

3. Auxiliary fans will recirculate air when the volume of air passing through the entry is not in excess of that passing through the auxiliary fan, and they may recirculate air when the current is in excess of that passing through the fan. Auxiliary fans will always recirculate air when the fan is inbye the last open crosscut.

4. Booster and auxiliary fans are not desirable as regards either economical operation or safety when used to offset the defects of leaky stoppings and doors, or as a substitute for properly maintained air courses.

5. Booster and auxiliary fans driven by electricity have ignited gas and have caused mine fires and gas explosions, resulting in loss of life and property.

6. In the event of a mine fire a booster fan may interfere seriously with or may aid in getting the fire under control.

7. When ventilation is dependent upon a booster fan and at the time of an explosion the fan cannot be operated by reason of being wrecked or the power being shut off, the recovery of the mine is seriously retarded and the hazards of the rescue crew and of any survivors of the explosion are greatly increased.

POINTS UPON WHICH OPINIONS MAY DIFFER

The points on which there may be differences of opinion are:

1. Whether booster fans and auxiliary fans can be so placed and equipped with explosion-proof motors that there will be no danger of gas ignition by the motor or by static electricity from the belt or fan.

2. Whether storage battery power, compressed air, or some form of power other than line electricity can be used safely.

3. Whether booster fans can be so protected that the possibility of mine fires is eliminated.

4. Whether underground fans can be so equipped as to be proof against conditions that might interfere with continuous operation to the same degree as outside fans.

5. Whether the use of automatic starters on underground fans can be considered a safe practice.

6. Whether the same protective devices can be applied to underground fans that are required by law on surface fans, such as offsetting the fan from the line of an explosion force, the use of explosion-relief doors and of a recording pressure gage, and in case of stoppage or slowing down the flashing of a light or sounding of a gong at some point where there is a constant attendant, as in the engine room.

DANGERS OF UNDERGROUND FANS

I will leave these points for discussion and will cite some of my own observations on the use and dangers of underground fans, together with some information received from others.

Ventilation in mines is provided for three general purposes: (1) The prevention of injury or death by burning or from coal-dust explosions augmented by methane; (2) the prevention of the loss of efficiency in employees which results when the oxygen content of the air drops below 20 per cent., and (3) the protection of the health of employees from the exudations from the lungs and body, as air should be sufficiently active to quickly break up, diffuse and carry away these exudations.

Booster fans in coal mines in the United States are, in my opinion, a direct hazard and too often a sign of weakness in the system of ventilation. This weakness may be due to a change in plans whereby a mining operation may be extended several times the distance originally contemplated, without the retention of sufficient pillars in which the additional entries required for proper ventilation may be driven, or it may be due to a day-to-day method of mining in which the aim is to keep down the daily cost regardless of future air requirements.

The installation of auxiliary fans is an attempt to make the mine safe from gas ignition at a cost less than that required to course the air by erecting board or canvas line brattices. The use of auxiliary fans may be inspired by false economy, by operating plans which increase distances between breakthrough, or by a failure of the ventilating system from leaky stoppings and clogged air courses. The stoppings which permit the greatest leakage and consequently favor recirculation of air have

been described² by the U. S. Bureau of Mines. Board stoppings and gob stoppings permit the greatest leakage, and monolith concrete the least.

My observations have been that the responsibility for the installation of booster fans and in many mines of auxiliary fans lies in leaky doors, stoppings, and overcasts; in the use of too many doors in lieu of overcasts and the substitution of canvas, board or gob for concrete stoppings in crosscuts; and in restricted air courses and air courses closed or partly closed by falls of roof or by heaving bottom. The cost of correcting these conditions may be greater than the cost of installing a booster fan, but the loss of mine safety incurred by failure to correct them is too high a price to pay. A combination of these conditions is not unusual. The restricted air courses may be due to undersized overcasts, right-angle turns, pack walls along the entries, small airways, and airways restricted by falls of roof or by cars; sometimes by timber or other supplies.

Booster fans may also be installed on the initiative of the company or on instructions from the mine inspector to increase the amount of air or where an increase in air circulation is so urgent that time will not permit the cleaning of the air courses and the building of tight stoppings. A change of management or an unexpected amount of methane may initiate the installation of booster or auxiliary fans.

Where air splits are of unequal length booster fans may be installed on the split of the greater resistance in lieu of building up the resistance on the shorter split by means of a regulator.

Another condition which is rare in the United States but may occur in Nova Scotia and Great Britain is that of coal so thin and air courses so long that with tight stoppings and clean air courses the resistance is so great that the air cannot be forced through in sufficient quantities at ordinary pressure to keep the oxygen above 20 per cent. and the percentage of methane below an amount that can be detected with a miner's safety lamp. Under such circumstances the driving of additional entries or the enlargement of the air courses by brushing the roof will greatly extend the time before the installation of booster fans must be considered. The additional entries should be driven and the roof brushed as the entries advance, rather than waiting until the demand is acute and the cost of doing all the work at one time looks prohibitive.

If the readers of this paper will visualize the mines using booster fans with which they are familiar and will eliminate all mines where there are wooden stoppings or overcasts or some other form of leaky stoppings, they will have eliminated a large percentage of the mines using booster fans. If they then eliminate all the mines where the air courses are restricted in driving or are restricted by pack walls, heaving bottom, falls of roof, or

² R. Y. Williams: Mine-ventilation Stoppings with Especial Reference to Coal Mines in Illinois. *Bull.* No. 99, Bur. Mines. (See p. 16 for computations made and summary of costs of erection and maintenance of stoppings.)

undersize overcast, they will have eliminated practically all the mines in which booster fans are used.

RESTRICTED AIR COURSES

Recently my attention was called to a mine where at a cost of only \$2500 an entry could have been enlarged so as to effect a reduction of the water gage 1 in. on 200,000 cu. ft. per min. In another mine a like reduction in pressure might have been obtained on 150,000 cu. ft. at a less cost. The first job involved enlarging a short piece of entry, and the second eliminating several bad angles in a short section of an air course. The saving in power effected by correcting these restricted airways would have soon paid for these jobs.

A certain mine operating under Government lease encountered methane, and a number of underground fans were installed. A change of personnel in the operating company resulted in cleaning the air courses from the surface, replacing the leaky stoppings with concrete stoppings, and enlarging the overcast. All the underground fans, except those used in electrical stations, were then removed. After this improvement the fire bosses seldom found any methane, while there had been daily accumulations to be moved when the auxiliary fans were in use.

Another company holding leased Government land became much concerned over the amount of methane reported in its mine. Underground fans were first installed, but after a change of personnel five concrete stoppings and one overcast were erected, and the conditions were so much improved that these fans were removed and the methane was reduced to a safe amount.

COMPARISON OF INSTALLATION OF UNDERGROUND AND SURFACE FANS

It is freely admitted that the ventilation of coal mines should be continuous except during necessary fan repairs and that should the fans be stopped even for only a short period all employees should be withdrawn from the mine. This practice is generally followed by coal companies and in some States is prescribed by law. Outside fans for ventilating coal mines are equipped with many commendable devices to give a warning by light or sound when the fan speed or air pressure has been reduced below a prescribed minimum. Recorders are used so that the operation of the fan can be checked for any time throughout the year. One Colorado company uses automatic devices to open and close doors when one of the two surface fans stops, so that the other fan will assume the burden of ventilating the mine until the men are withdrawn. Devices are used to give warning when the bearings of the fan reach a certain temperature. Surface fans are offset from the line of a possible explosion and are equipped with explosion-relief doors.

"Out of sight, out of mind," appears to be the slogan for underground fans. If recording and safety devices of the type mentioned are of value

on outside fans they are even more essential on underground fans. To date, however, I have failed to observe any underground fan offset so as to prevent its injury in the event of an explosion, although G. S. Rice has shown me a sketch of such an arrangement at a mine in Nova Scotia. Underground fans are not equipped to flash a light in the engine room when the fan has slowed down to a dangerous degree or to indicate a heated bearing, nor are they equipped with a recording device to show their performance throughout each 24-hr. period. Separate electric circuits unaffected by haulage motors, mining machines, or other underground motor-driven appliances are not universally used to insure continuous fan operation. Unless underground fans are continuously operated and are equipped with the same warning and safety devices as outside fans, they should be condemned as vigorously as one would condemn a surface fan under such circumstances. In fact, I am not sure that the laws of some States are not violated when a booster or auxiliary fan is installed without the same provisions as are provided for surface fans. One alternative is an overseer stationed at the fan to care for it and to warn and withdraw the miners in case of danger.

MOVING METHANE IN A BODY

Records indicate that by reason of intermittent operation, booster fans have moved bodies of gas to some distant point, where they have been ignited. This hazard is greater than that involved in practice, universally condemned, of moving gas by brushing with a coat, shovel or water spray. One of the readers of this paper has suggested a subheading on motorizing gas brushing or applying mechanically in intermittently operated fans the principle of the old coat and shovel method of moving gas. Underground fans that are not operated continuously are not alone responsible for moving bodies of gas, as bodies of gas have also been moved and ignited at a distance as a result of the starting of surface fans or of the recouring of air by the erection of doors, stoppings, curtains and line brattices.

A few instances of bodies of gas being moved considerable distances without being broken up and diffused below the explosive limit are cited below.

About 10 years ago I reviewed a report³ on a mine explosion in which the conclusion was expressed that a body of gas leaked through and accumulated in front of a stopping while the surface fan was idle and undergoing repairs. After the fan was started several men entered the mine on the intake, and at a distance of 1500 ft. beyond the place of accumulation they lighted the gas, originating an explosion. It was difficult at first to understand where this gas came from until a small

³ H. I. Smith and R. J. Hamon: Methane Accumulations from Interrupted Ventilation. *Tech. Paper* 192, Bur. Mines, 10.

body of gas was found in a cavity in front of a leaky stopping. The velocity of the air was then determined and multiplied by the number of minutes between the time of starting the fan and the explosion. The movement of this accumulation of gas without breaking up checked closely with the time the fan had been operating and the distance the gas had traveled.

In another instance reported to me by G. S. Rice, after an inspection and a report that the mine was clear, the fan was reversed and a body of gas in a caved area traveled approximately 1300 ft., where it was ignited by men with open lights. On one occasion while investigating a mine explosion after the fire boss had reported everything clear and men were erecting brattices with open lights on the return, I walked into a body of gas on the main entry. This gas had traveled several hundred feet from inaccessible workings to the point where my light was extinguished. The movement of the gas was due to reestablishment of the ventilation by the construction of brattices. In another instance the mine foreman went into a mine to clear an entry of gas and start a booster fan. He telephoned to the surface to throw in the power, and within a few minutes the switch breaker was thrown out, thereby indicating the time that the explosion occurred. The fan was on the intake 300 ft. from the face, and the time from throwing on the power until the switch breaker was thrown out was very close to the time required for gas at the face to be moved back to a point opposite the fan. Presumably the gas was sucked through leaky stoppings and ignited by the motor; the foreman carried a safety lamp but no open light. The air course in this mine was badly restricted by pack walls. There are many other instances of gas having moved in a body after the circulation was reestablished and ignited at a distant point.

It is apparent that in places containing small amounts of methane at the face the methane percentage gradually increases after the air current is cut off,⁴ and even after the current is reestablished there is a continued increase in methane at the face for a period of time. In view of this continued increase and the fact that methane may be moved in an explosive body for long distances from the point where it may accumulate, the use of automatic starters on underground fans is decidedly dangerous. Intermittent supervision of underground fans is likewise dangerous.

LEAKY STOPPINGS, DOORS, OVERCASTS AND PARTITIONS

Twelve years ago my attention was first drawn to recirculation. I made a test on a booster fan and found that in the return there was but 13,000 cu. ft. of air while at the booster fan there was 29,250 cu. ft.

⁴ H. I. Smith and R. J. Hamon: Methane Accumulations from Interrupted Ventilation with Especial Reference to Coal Mines in Illinois and Indiana, *Tech. Paper* 190, Bur. Mines.

Measurements were then made of the air on the haulage entry opposite the fan, showing 29,200 cu. ft. of air. The loss of 16,200 cu. ft. of air occurred within a distance of 2000 ft. and indicated approximately 500 cu. ft. per stopping and door. There was more than 0.5 per cent. of methane in the air. In my report attention was called to the possibility of igniting the gas by static electricity, even though a protected motor may have been used to drive the fan. I can cite only one instance⁵ where an ignition is attributed to static electricity in a coal mine. The static electricity is credited with accumulating on the blades of a booster fan driven by compressed air. Many grain-dust explosions have been attributed to static electricity by D. J. Price.⁶ The chain dangling on the rear of all gasoline trucks to discharge the static electrical accumulation on the tank is familiar to everyone.

A small fan set on the surface at a certain coal mine was used to ventilate a shaft before underground connections had been made to the air shaft. The shaft was divided by a wooden partition, and as it was found that the air current was too sluggish to keep the mine clear of methane a second or booster fan was set at the bottom of the shaft to increase the circulation. The additional fan did not help much in ridding the development entries of gas, and after checking up carefully with an anemometer it was found that although some 10,000 cu. ft. of air was being handled by the first fan and a similar amount by the fan at the shaft bottom there was no movement halfway down the shaft, or at least insufficient movement to turn the anemometer. Investigation showed that the air handled by the top fan was leaking out before it reached the bottom of the shaft and that the air handled by the bottom fan was being pulled through the brattice boards and recirculated. The use of a large canvas tubing connected directly to the top fan resulted in providing more fresh air than the top and booster fan combined had been furnishing, and accordingly the booster fan was eliminated.

The results of a test as set forth in Table 1 show the air losses in an entry where all the stoppings built 30 months previously had been torn out and new wooden stoppings plastered with mud had been put in. They may be considered as indicating a minimum in the matter of recirculation where stoppings of this nature are used under a like pressure.

The figure of $1\frac{1}{2}$ c. per hp.-hr. was given to me by the engineer as representing the actual horsepower cost at this mine. The conclusions drawn from this table are that wooden stoppings should not be used when the expected life is over 18 months to 2 years nor when the water-gage pressure exceeds 0.2 inch.

Sometime ago in a mine where plans had been made for a new fan I measured 2700 cu. ft. of air leakage through 2 doors and 4 stoppings, or

⁵ J. A. B. Horsley: *Electrical Installations in Mines. Iron & Coal Trades Review*. (Dec. 17, 1926) **113**, 940.

⁶ Engineer in charge, development work, Department of Agriculture.

TABLE 1.—*Air Losses in Entry Where All Stoppings Had Been Torn Out and New Stoppings Plastered with Mud Put in*

	Air, Cu. Ft.	Water Gage, In.	Air Loss, Cu. Ft.	Theoretical Loss per Hp.-yr. at 1½ C. per Hp.-hr.	Loss at One-third Fan and Motor Efficiency	Loss per Stopping per Year
At fan.....	31,700	0.8				
Distance from fan:						
Ten stoppings.....	27,000	.35	4700	\$247.36	\$742.08	\$74.21
Twenty stoppings.....	25,000	.25	2000	48.09	144.27	14.43
Thirty stoppings.....	23,000	.15	2000	43.56	130.68	13.07

450 cu. ft. each. The pressure through the stoppings as registered by the water gage was 0.4 in. In another mine 6500 out of a total of 11,100 cu. ft. on the split leaked through 14 stoppings and 4 doors—a total loss of 58 per cent., or 3.2 per cent. through each crosscut. In another, 2300 out of 6800 cu. ft. leaked through 5 stoppings and 2 doors. In a mine where considerable concern was manifest over the amount of methane present a loss of 39,000 out of 54,000 cu. ft. being handled by the fan occurred in the temporary fan housing before the concrete housing was started. In another mine, as reported to me, 65,000 cu. ft. was lost through the fan housing and wooden partition in a shaft before the air reached the shaft bottom. A loss of 10,000 out of 50,000 cu. ft., or 20 per cent., was observed in another fan housing that had been serving the mine for a number of years. A booster fan showed a leakage of 62 per cent. through two doors and three brattices under a water-gage pressure of 0.9 in. Leaks of this kind if not corrected lead to the use of booster fans, and such leaks are often needed to make a booster work in a manner to appeal to the mine foreman.

Some mining companies have ventilation engineers on their pay rolls, and there is much for specialists of this kind to accomplish. If consulting mining engineers were called on more generally to solve ventilating problems the cost of many booster and auxiliary fans, as well as many surface fans, would be saved, and better ventilation would result. In some of the examples above cited it would have required but one or two concrete overcasts and a few concrete stoppings to eliminate a large part of the leakage. I have observed some very bad air losses where temporary stoppings were used on development work before a change was made to permanent stoppings. The temporary stoppings, which were forgotten, were a source of leakage of considerable magnitude. In one such case 3800 out of 19,100 cu. ft. leaked through one door and one stopping at a water-gage pressure of but 0.75 in. The regulator formula,

$A = \frac{0.0004q}{\sqrt{i}}$, shows an opening in these two crosscuts equivalent to 1.76 square feet.

In contrast to these losses there are plenty of mines where air losses are reduced to a minimum, and the examples cited above should be considered exceptions, although I have not mentioned nearly all the large losses which I have in my notes but there is enough to show that air losses are a problem for closer observation and bear directly on the use and dangers of booster fans. In every example of air loss I have cited either underground fans were in use or efforts were being made to increase circulation by speeding up or figuring on new surface fans.

RELATION OF BOOSTER FANS AND LOW OXYGEN

To illustrate an acute danger in the use of booster fans, I would quote Daniel Harrington,⁷ who observed a case where workers receiving air passing through a canvas tubing had difficulty in keeping their carbide lights burning. Matches would not burn at the fan, yet 50 ft. outside the fan a match would burn brightly, thus indicating the extent to which the booster fan failed to draw on fresh air but 50 ft. away. Mr. Harrington has also shown me a long list of fires and explosions resulting from booster fans, including one where an ordinary office fan used to clear entries of gas resulted in casualties and others involving longwall mining where men have been burned by the ignition of gas at the fan motor.

A booster fan in a certain coal mine picked up 7600 cu. ft. of air in passing 28 stoppings, or an average of 270 cu. ft. per stopping. No doubt most of this was sucked through the 10 stoppings nearest to the fan. In the same mine black damp drove the men from their working places when the booster fan stopped. The air recirculated by this booster fan contained less than 18 per cent. of oxygen.

In another mine where the oxygen in the ventilating current was less than 18 per cent. it was found that the booster fan on the main intake actually obstructed the air course, for less air passed through the air course after the booster was installed and in operation than before. By opening the doors on both sides of this disk fan a slight increase in the amount of air over that supplied by the surface fan was obtained. The two other boosters circulated more air than the one on the main intake. The turn in the low-oxygen section was usually less than in the rest of the mine, causing a reduction of tonnage from the entire mine. The cars pulled per driver were but one-half a normal day's run, and only with difficulty was it possible to get the loaders to work a full shift. In spite of this the men appeared to feel that as they were getting a strong current the air was all right, but they could not do the work contemplated. Within a few weeks after analysis of the air was received and

⁷ Consulting mining engineer, Salt Lake City, Utah.

condition of the air known a complete change was made and the trouble was remedied by sinking a new air shaft. After the condition was remedied one driver was eliminated and the difficulty in keeping a uniform turn throughout the mine was reduced to a normal operation.

In still another mine the outside fan delivered 35,000 cu. ft. of air per min. while the booster fans underground circulated 53,000 cu. ft. Analysis of the air circulated by the underground fans analyzed from 17 to 19 per cent. of oxygen.

According to Mr. Garcia's report⁸ at the Pittsburgh meeting of the A. I. M. E., 200 cu. ft. of air per man is the maximum required in any State. At this figure, a mine with 500 men would require but 100,000 cu. ft. of air, but it is not uncommon to see a fan deliver double or triple this amount of air for fewer men. The question is, what becomes of the remainder? This question is partially answered in the operating regulations as applied to coal leases on public lands, where but 100 cu. ft. of air per man is required in the last open breakthrough. This is equivalent to 300 to 1000 cu. ft. per man measured at the intake, for to get 100 cu. ft. of fresh air per man at the last breakthrough the stoppings must be tight and the air courses clean and unrestricted. Under these regulations booster fans are not permitted to recirculate more than 10 per cent. of the air. These stipulations will largely eliminate the dangers of low oxygen and leaky stoppings, if in fact they do not also eliminate the booster fans, for booster fans are seldom if ever found where leakage is as little as 10 per cent.

Today there are no booster fans on leased public or Indian lands. There are four blower fans used to ventilate underground hoist rooms. The lessees who have improved their ventilation and eliminated the necessity for booster and auxiliary fans are well pleased with the improved conditions.

PRESENT TENDENCIES TOWARD UNDERGROUND FANS

A remarkable contrast is shown in the description of the New Orient mine,⁹ and that of the Nemaacolin mine.¹⁰ The first shows a four-entry system and reports the use of 57 blowers or auxiliary fans. The second shows seven main entries, three intakes and four returns, and all working places are ventilated by line brattices. This contrast is referred to in order to emphasize the extremes in present-day practices in mine ventilation. Personally I much prefer the use of the additional entries and line brattices to the auxiliary fans.

⁸ John A. Garcia: State Coal Mining Laws Concerning Ventilation. *Trans.* (1926) 74, 409.

⁹ New Orient Mining Methods. *Coal Age* (Sept. 9, 1926) 346.

¹⁰ W. Z. Price: Operation of Nemaacolin Mine. *Trans.* (1926) 74, 559.

There are places such as in underground pump stations, hoist rooms and electrical stations where the use of auxiliary fans and canvas tubing in place of line brattices appears to be appropriate, temporarily at least, but power is always uncertain at the face, and when a fan stops even for a short time there is no way in which the main fan can take care of the gas accumulations. In some mines the power goes off so often that a fire boss cannot withdraw all the men and make an examination every time it happens. When the main fan stops the men, including those at the face as well as those at the shaft bottom, are withdrawn, and it is of course particularly important to withdraw those in the more dangerous parts of the mine should there be any interruption of the ventilation from any cause.

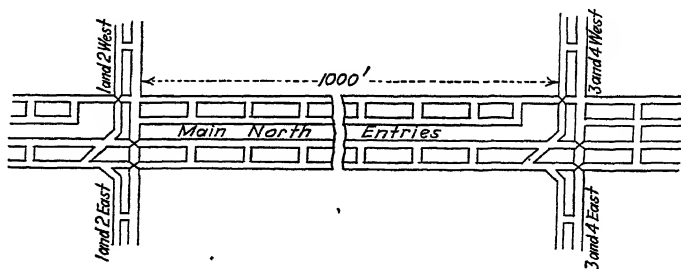


FIG. 2.—SOLID PILLAR PLAN USED IN DEVELOPING NEW COAL MINES IN INDIANA DURING 1917-1918.

One way to eliminate the dangers of booster and auxiliary fans is to eliminate the necessity for them by getting a large percentage of the air from the main fan to the face with a minimum of power. This can be done in a number of ways, such as sinking shafts in advance territory, or erecting all stoppings and overcasts of concrete and eliminating doors. I was instrumental in bringing about the adoption, in several mines opened up in Indiana in 1917 and 1918, of a plan having in view more economical ventilation. This plan provides for two pairs of main entries with a solid pillar of coal between the two pairs. One pair is used for intake air and one pair for return air; the solid pillar is to be cut only at the cross entries. The details of the plan to make the most economical development under the Indiana mining scale were worked out by the chief engineer of the company. The same scheme has been more recently followed in large mining operations in Illinois.

Fig. 2 shows the Indiana or solid pillar system which effectively eliminates the booster-fan problem by reducing the air leakage to a minimum and thereby eliminating the conditions that call for the installation of a booster fan. The demand for auxiliary fans or blowers can also be greatly reduced by driving more entries, building more concrete

stoppings, and using fewer doors. The aim in ventilation should be to get to the last crosscut 90 per cent. of the fresh air handled by the surface fan, or nine times the amount to be found in some mines.

Before any surface, booster or auxiliary fans are installed in or at a coal mine fan houses and airways should be examined and it should be found out whether the appropriation should be made for a fan or for changes in the mine. If there is not a ventilation man in the organization advice should certainly be bought from a consulting engineer familiar with up-to-date coal mining ventilation who can point out the spots in the mine that are below par.

DISCUSSION

R. D. HALL, New York, N. Y.—I am not a strong advocate of booster fans. Usually they can be made unnecessary by regulating the main current, cleaning and enlarging the roadways and making the stoppings tight.

The auxiliary fan with its tubing rests on an entirely different basis. It takes the place of a door or curtain which is used to deflect the main current and to compel it to travel to the face of a room or entry. In consequence its safety and efficiency must always be compared with the same features as afforded by doors and curtains.

If an auxiliary fan is run continuously it is just as safe as any other method of ventilating a limited area of working face provided there is no recirculation and the gas is not permitted to pass a non-permissible motor as it leaves one place to go to another.

Wherever possible crosscuts should be eliminated. The auxiliary fan provides a means of delivering air long distances without putting in crosscuts. When crosscuts are set 100 ft. apart and the pillars to be crosscutted are 100 ft. thick, the men are likely to be 200 ft. away from the air where line curtains are not used. Surely there is no question as to the desirability of a fan and tubing that take air from the main current and circulate it to men who would otherwise work in stagnant devitalized air.

The problem should be regarded from an engineering point of view. An engineer would not say that if there is danger in a certain form of operation it should not be used. Finding it advantageous and helpful he would inquire how it may be made safe and finding the way would proceed to use it in the safe manner. Most engineering developments bring problems of safety which have to be studied and met.

J. J. RUTLEDGE, Baltimore, Md.—There was a solid pillar in Mines 6 and 8, Monogah, W. Va., in December, 1907, at the time the explosion occurred in these mines. It was of very material assistance in restoring ventilation after the explosion. A method of working involving the use of continuous, solid, unbroken main-entry pillars is shown in the proceedings of the Chicago meeting of the Mine Inspectors' Institute of America.¹¹

D. HARRINGTON, Washington, D. C.—Mr. Hall has presented the advantages of the auxiliary fan, but one might conclude that with the auxiliary fan in use in the mines doors or brattices would be unnecessary. Also, that if auxiliary fans were used all dangers from doors and brattices would be eliminated.

The worst effect of the auxiliary fan is that it gives to the mine operating official, the foreman, the first boss, probably the superintendent, the idea that by its use he

¹¹ Walton Rutledge: Description of Plan for Coal Mine Which Gives Off Fire Damp or Inflammable Dust. *Proc. Mine Inspectors' Institute of America* (1910) 146.

can make up his delinquencies in other lines, and this is borne out time and again when you go into mines where they have auxiliary fan installations.

I do not know of a single instance in the United States, where the auxiliary fan is run 24 hr. in the day. That means that if the faces which are being ventilated give off gas there is a gas accumulation for every auxiliary fan in the mine every day in the year, and that is what you actually get.

E. L. COLLIS, CARDIFF, Great Britain.—The main problem receiving thought is that of explosions, the presence of methane in the air. Perhaps a little less attention is paid to the fact that the miner underground will find air with 20 per cent. oxygen, with enough methane to make it an explosive mixture, perfectly healthy if that atmosphere is moving sufficiently. On the other hand, he may find himself in a stagnant and depressing atmosphere which a mining engineer is perfectly satisfied with because it is not explosive.

I mention this point because it seems to me the question of the auxiliary fan comes in here, and even recirculation of a stagnant atmosphere, which is not explosive, may be very beneficial to the miner at the front. Even if it is explosive, so far as he is concerned, his physiological condition being satisfied by the fact that the air is in movement is a very important point when you are dealing with deep and hot mines. It is a matter which is receiving some attention in Great Britain.

D. HARRINGTON.—That particular phase of the question is not taken up because there are very few places in the mines of the United States where the physiological factor enters into it, as far as high temperatures are concerned. Very few coal mines in the United States have temperatures as high as 60°F.

C. EVANS, JR., SCRANTON, Pa.—I beg to differ with you. In the mines I have charge of we have a great many situations where we are attempting to reopen caved ground. It is very difficult to maintain the return air system in such ground and we are coming more and more to use the auxiliary fan. That is entirely on account of physiological conditions. There is no danger from gas. The mines were bled free of gas years ago. We are putting in booster fans in order to take away the products of decaying wood.

In one place the temperature was so high that we recorded it over a period of 2 weeks and noted as high as 94°F. That particular situation was in a bed that was only 120 ft. below the surface. It was covered with a broken shale roof and 60 to 70 ft. of boulders and clay mixed together, which apparently formed a blanket that held all the heat that was generated by the movement of the strata and the decay of the enormous masses of timber. The bed at that point was 25 to 30 ft. thick.

I have no records of humidity, but it was very high. The conditions in there were almost intolerable. I was exhausted at the end of 15 or 20 minutes' walk through the gangways.

H. I. SMITH.—Have you taken any analysis for oxygen? Is it less than 20 per cent., and is the loss of efficiency due to low oxygen?

C. EVANS.—We made no analysis for oxygen.

H. I. SMITH.—You will find it very interesting if you do.

C. EVANS.—The region was economically unworkable, so we gave it up after a short time. We are now preparing to strip it.

F. HAAS, Fairmont, W. Va.—I am already on record on the auxiliary fan and will not repeat. I agree with Mr. Hall that from an engineering standpoint it offers pronounced advantages and for certain emergencies is essential. In our mining experi-

ence we have no use for booster or auxiliary fans in the usual or ordinary operation of mines. An objection to the booster fan not previously mentioned is that it is of too limited capacity to take care of the wide variation in the requirements of some gaseous mines. From an operating standpoint I would not tolerate a booster fan because of its limitations and liability to misuse.

Mr. Smith has made the statement that 90 per cent. of the air should reach the last crosscut. A statement like this, if unchallenged, sets up a standard. Standards are good because they give the operating man who has not had the experience some idea of good practice.

In my opinion Mr. Smith has set his standard entirely too high. In our experience we consider it good practice if we deliver 80 per cent. of the total air to the entrance of the split. Within the split we are satisfied if 50 per cent. of the air passes the last crosscut. This does not mean that 50 per cent. of the air is lost or unaccounted for as it is neither necessary nor desirable for all the air to pass the last crosscut.

C. E. KREBS, Charleston, W. Va.—There is only one use that we could make of the auxiliary fan and that is in driving entries. In developing an area of coal that we wish to develop rapidly, by working with a tube and the auxiliary fan and only using a few men, we would eliminate the crosscuts and the necessity of building brattices. We would also have more efficiency in the entries in carrying the air forward to the mine workings later on, so that there would be crosscuts only every 300 ft., instead of every 50, 80 or 100 ft., as the law requires in different states.

I think Mr. Haas is correct in his assertion that 50 or 60 per cent. air efficiency at the face of the entries is about all that it is possible to get however good the breakthrough may be, because there are always leakages where the air is lost in different breakthroughs. I agree, too, with Mr. Haas on booster fans. They should not be used in mines in any place if we can possibly do without them, and in good, regulated mines it is well to do without them, because better ventilation is required than booster fans can give.

J. J. RUTLEDGE.—Mr. Haas, please repeat that standard water gage per 100,000 cubic feet?

F. HAAS.—Sixteen-hundredths of an inch. The fan is producing 325,000 cu. ft. With a 1.7-in. water gage. That would be equivalent to 100,000 cu. ft. of air with about 0.16-in. water gage.

C. EVANS.—Was it in actual operation? You are not reporting a calculation?

F. HAAS.—The 0.16 in. per 100,000 cu. ft. of air is a calculation. The 325,000 cu. ft. of air with 1.7-in. water gage is a continuous operation.

H. I. SMITH.—I appreciate the figures presented by Mr. Haas giving the percentage of air reaching the various splits and last breakthrough for that group of mines. The figures are above the average. It was not my intention that 90 per cent. should be set as a standard. A standard should be a percentage which may be economically obtained in an average mine. It is hoped that additional figures will be presented in discussion as this is an important economical and safety problem on which there is too little available information and a problem on which the Mine Ventilation Committee could devote some time to the advantage of mine operators and their ventilation engineers.

F. HAAS.—It is not a matter of compromise; it is a matter of fact.

H. I. SMITH.—I agree with Mr. Hall that the use of doors and curtains for doors has hazards besides being broken down. Curtains and doors should be eliminated as far as possible. There is a difference between auxiliary fans and line brattices or doors. Line brattices and doors are sometimes out of commission and cannot move the gas. Auxiliary fans are out of commission at all times when the power is off and the number of auxiliary fans is usually much greater than the number of brattices or doors that may be out of commission.

R. D. HALL.—I hope the time will come when auxiliary fans will be driven by batteries. Then we will not be troubled by power failure.

Sources of Dust in Coal Mines*

By J. J. FORBES,† PITTSBURGH, PA. AND ALDEN H. EMERY,‡ WASHINGTON, D. C.

(New York Meeting, February, 1927)

THE data contained in this paper were collected during the course of an investigation which covered 15 representative coal mines in six coal-mining states. The purpose of the investigation was to determine by atmospheric sampling how much dust was raised into the air during the various operations of coal mining, such as undercutting with machines, loading coal by hand and transporting it from the working face to the surface. The data thus obtained were considered from two angles; namely, the possible physiological effect of inhalation and the explosion hazard of the dust. No attempt will be made to discuss the former factor in this paper.

CONDITIONS OF MINING

The mines in which this investigation was conducted were considered representative, and it is believed that the results obtained will suffice for the average dustiness encountered in most coal mines. All of the coal mined was bituminous, from beds 30 to 120 in. thick. Only the ordinary room-and-pillar system of mining was employed. Daily production ranged from approximately 700 to 1500 tons and the men employed from 150 to 450. Coal was undercut by shortwall mining machines in all mines, with the exception of one in which pick mining was employed. Loading at the face was done by miners; none of the mines used mechanical loaders. Electric drills, both for brushing roof and for shooting coal were commonly used, although some roof work was done with jack-hammers and a little of both with augers. Coal was hauled by motor, mule, or rope power in end-gate cars with capacities ranging from 1800 to 3600 lb. With few exceptions, coal was won at the face by dry methods, that is to say, no water was employed for wetting down the dust either during undercutting, loading or transportation.

OBSERVATIONS OF SOURCES OF DUST

The common sources of coal dust as observed, arranged by decreasing amounts, were as follows: (1) Undercutting with mining machines; (2)

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† Supervising engineer, Instruction Section, Safety Division, U. S. Bureau of Mines.

‡ Associate geologist, U. S. Bureau of Mines.

loading coal at face of workings, and (3) transportation of coal. The dust raised into the air at the face during undercutting varied from light to very dense clouds which made visibility poor. Very light to comparatively dense clouds of dust were raised during loading. In all instances the air was clear to vision during the transportation of coal.

COLLECTION OF DUST SAMPLES

All dust samples were collected to be as nearly representative as possible of the dust-producing operations, such as undercutting across a



FIG. 1.—APPARATUS USED IN COLLECTING DUST SAMPLES. COLLAPSIBLE STAND MAY BE ADJUSTED TO ANY CONVENIENT HEIGHT.

10, 20, 30 or 40-ft. face, completely loading cars by one or two workers, and transporting coal in rapid or slow-moving trips. For undercutting, the collection was commenced before sumping operations, and was continued until the entire face was undercut. Samples were taken in development entries and along main haulage-ways, where mules, motors or ropes were used. None were taken in pillar workings. In order to get the average dustiness throughout the day, dust sampling was done

before men started to work, and throughout the working shift at intervals of $\frac{1}{2}$ to 1 hr. These were supplemented by atmospheric dusts taken at representative places throughout the mine.

The sampling outfit used in this investigation was the impinger.¹ (Fig. 1.) This instrument consists of a small portable hand-cranked suction pump connected by means of a flexible rubber tube to an 8-oz. wide-mouth bottle containing a rubber stopper bearing two $\frac{1}{2}$ -in. glass tubes, one of which is the attachment to the suction pump and the other the inlet for the dust-laden air. The latter is the impinger proper; it consists of a glass tube drawn down sharply at one end to a small orifice and supporting a brass plate 5 mm. from the opening, and the whole is immersed as deeply as possible in 150 c. c. of suitable liquid, water usually. The pump and bottle are both supported on a collapsible metal stand at a comfortable working height.

COMPOSITION OF DUST SAMPLES

The composition of the dust, as determined by petrographic examination, varied with the material handled. Mechanical drilling was done in sandstone, shale and slate roof. Dust samples collected during these operations showed little or no coal. Auger drilling by hand was performed in both roof and coal, and the composition of the dust raised consequently corresponded to the material drilled. Mining machines occasionally dug into the floor, but more than 99 per cent. of the dust raised during undercutting was coal. Bony partings shot down with the coal occasionally contaminated the dust produced during loading, but 99 per cent. of this material was coal dust. Along the haulageways the composition of the dust varied. In non-rock-dusted entries, the suspended particles were almost entirely coal but in rock-dusted places some of the atmospheric dust was of the same material as that used for rock-dusting.

DUSTINESS BY OPERATIONS

Table 1 gives by count and weight and in order of decreasing magnitude of total counts, the amount of dust raised per unit of volume of air during mining operations and includes jackhammer drilling, undercutting coal in wide and narrow work, under both dry and wet conditions, drilling with electric drills and hand augers, loading coal at working face, picking at face and transportation of coal by mule, motor or rope haulage. In addition the table gives the amount of dust raised at the tippie after the coal has reached the surface.

¹ S. H. Katz and others: Comparative Tests of Instruments for Determining Atmospheric Dusts. *Public Health Bull.* No. 144 (1925) 62.

TABLE 1.—*Summary of Dustiness by Operations*

Operation	Number of Samples	Kind of Material	Millions of Dust Particles per Cubic Meter ^a of Air			Milligrams ^b of Dust Particles per Cubic Meter of Air		
			+10 Microns ^c	-10 Microns	Total	+10 Microns	-10 Microns	Total
Drilling:								
Jackhammer.....	6	Sand Shale	1,881	130,138	132,020	1,753.0	1,045.0	2,799.0
Electric.....	4	Shale	318	15,347	15,665			192.3
Undercutting coal, dry methods:	42	Coal	245	9,565	9,810	254.6	103.9	358.5
Narrow work.....	(11)	Coal	(168)	(11,890)	(12,058)	(325.3)	(146.9)	(472.2)
Wide work.....	(31)	Coal	(273)	(8,740)	(9,013)	(229.5)	(88.7)	(318.2)
Auger drilling by hand...	5	Coal Shale	25 $\frac{1}{2}$	1,959	1,984	29.8	16.3	46.1
Loading coal.....	40	Coal	41 $\frac{1}{2}$	1,500	1,541	66.6	28.1	94.7
Undercutting coal, wet methods.....	3	Coal	35	1,367	1,402	24.5	25.9	50.4
Tipple.....	4	Coal	18	1,159	1,177	23.8	7.9	31.7
Pick mining and loading coal.....	10	Coal	42	615	657	24.4	14.8	39.2
Loading rock.....	2	Sand Shale	6	291	297	6.6	1.7	8.3
Haulage:	70	Coal	4	248	252	3.9	3.6	7.5
Mule.....	(11)	Coal	(7)	(609)	(616)	(7.3)	(3.9)	(11.2)
Motor.....	(35)	Coal	(5)	(202)	(207)	(4.2)	(5.2)	(9.4)
Rope.....	(24)	Coal	(2)	(151)	(153)	(1.9)	(1.0)	(2.9)
Pick mining coal.....	6	Coal	20	148	168	16.0	5.7	21.7

^a 1 cu. m. = 35.314 cu. ft.

^b 1 mg. = about $\frac{1}{28,000}$ oz. avoirdupois.

^c Micron = about $\frac{1}{25,000}$ in. Throughout the paper +10 refers to particles more than 10 microns in diameter, and -10 to the particles below this limit.

Discussion of Data

(In connection with dust weights and counts in this report it is important to keep in mind that all the -10 micron dust, also probably all or nearly all of the +10 micron dust, in the tabulations in the report is of size which would very readily enter into an explosion.)

It will be noted that the most dust per unit of volume was raised by jackhammer drilling in roof; the average count of six samples from drilling at an angle of 10 to 15° up was 132,020 million dust particles per c. m. (cubic meter) of air. Electric drilling in either coal or roof ranked second in order of dustiness. Four samples taken from drilling at an angle of 10 to 15° up in roof gave 15,665 million dust particles per c. m. of air.

Dry undercutting in coal ranked third in order of dustiness, 42 samples, 11 being from narrow work (places under 12 ft. wide) and 31 from wide work (places over 12 ft. wide), averaged 9810 million particles per c.m. of air. The 11 samples, representing 118 ft. of cutting in narrow places showed an average dustiness of 12,058 million, and the 31 samples from 840 ft. of wide work an average of 9013 million. The fourth dustiest operation, namely, hand auger drills, operated in positions varying

from horizontal to vertical, showed 1984 million particles. Loading at face was the fifth in order of dustiness; 40 samples raised an average of 1541 million. Undercutting in coal using water on the cutter bar was sixth in total amount of dust, the three samples averaged showing 1402 million particles per c.m. air.

Pick mining and loading simultaneously was seventh in dust magnitude and showed an average of 657 million particles for 10 samples. Loading rock averaged 297 million for two samples; these were gathered in mines where much brushing was necessary to provide adequate height for haulage.

Transportation of coal from working face to tippie showed the least amount of dust of all mine operations except pick mining. Haulage by mule, motor or rope averaged 616, 207 and 153 million dust particles, respectively, and 252 million for the operation as a whole. Pick mining at the face showed the least amount of dust raised for any of the regular mining processes; the average for six such samples was 168 million.

The four samples taken around the tippie showed a large amount of dust raised by surface work, 1177 million particles for the four samples gathered.

The weight of dust particles ranged in decreasing amounts from 2799 mg. per c. m. of air for jackhammer drilling to 7.5 mg. for haulage. The average weight for "dry" undercutting in wide and narrow work was 358.5 and for loading coal, 94.7 mg. The operations with high counts of dust almost always showed correspondingly high weights and those with low counts showed low weights.

The above order of dustiness held in almost every individual mine, although the magnitude of the figures varied.

Dust Produced by Mechanical Drilling

Table 2 shows, by mines, the dustiness during mechanical drilling with either jackhammer or electric drills.

TABLE 2.—*Dustiness by Mines during Drilling*

Mine	Number of Samples	Millions of Dust Particles per Cubic Meter of Air			Milligrams of Dust Particles per Cubic Meter of Air		
		+ 10	− 10	Total	+10	− 10	Total
Jackhammer							
A	4	2,642	182,349	184,993	2,478	1,497	3,976
L	2	359	25,715	26,075	303	141	444
Electric Drill							
C	3	391	17,803	18,194			230.7
D	1	100	7,979	8,079	44.1	33.1	77.2

Discussion of Data

There was a wide range in the dustiness from jackhammer and electric drilling in different mines. Four samples collected in one mine while a jackhammer was drilling in sandstone, shale and slate roof in turn, at an angle about 10° up, the cuttings dropping $3\frac{1}{2}$ to 5 ft., in almost still air, show an average count of 184,993 million particles; whereas, in another mine where two samples were taken after jackhammer drilling in relatively soft shale and sandstone under similar conditions, only 26,075 million particles were raised. Similarly, in a mine where three samples were taken while electric drills were working at angles of 10 to 15° up, the cuttings falling $3\frac{1}{2}$ to 4 ft., in almost still air, 18,194 million particles were raised; and in another mine where only one sample was taken, under similar conditions, 8079 million were produced.

The weight of dust in mg. per c.m. of air for jackhammer drilling is very much greater than the weight for electric drilling. In each instance the weight parallels the dust count.

The wide range in dustiness in different mines during jackhammer drilling, and similarly for electric drilling, can be accounted for by the different nature of the material drilled, condition of the machines used, continuity of the operation, depth of hole, angle of drilling, distance through which cuttings fall, and air movement at points of sampling. The difference in weights of dust between the two methods of drilling is due to the difference in the two types of drills. It must be borne in mind, however, that while the amount of dust per unit volume of air raised by these operations is excessive, nevertheless, the total amount so raised is small because of the infrequency of such work. The physiological effect would be largely on the few men required to run these machines.

The dust produced by these activities contained very little coal. As all of this drilling was in rock roof, it was only as thin lenses of coal were struck that anything other than siliceous dust was raised. The composition of the atmospheric dust followed very closely the composition of the rock drilled.

Dust Produced by Undercutting Dry

Table 3 shows the dustiness of mine air while coal was being undercut dry (Fig. 2), and the number of samples averaged to give each figure, and represents the average dustiness per unit of volume of the working places at each of the mines during this one operation.

TABLE 3.—*Dustiness by Mines When Undercutting Dry*

Mine	Number of Samples	Millions of Dust Particles per Cubic Meter of Air			Milligrams of Dust Particles per Cubic Meter of Air		
		+10	-10	Total	+10	-10	Total
<i>F</i>	6	977	25,564	26,541	560.2	192.6	752.8
<i>M</i>	11	178	16,812	16,990	505.5	213.4	718.9
<i>D</i>	5	88	4,537	4,625	93.8	38.3	132.1
<i>E</i>	2	474	2,724	3,198	119.5	83.5	203.0
<i>K</i>	7	73	2,213	2,286	89.2	33.2	122.4
<i>L</i>	9	56	1,974	2,030	35.8	26.0	61.8
<i>H</i>	2	35	1,020	1,055	58.8	18.3	77.1

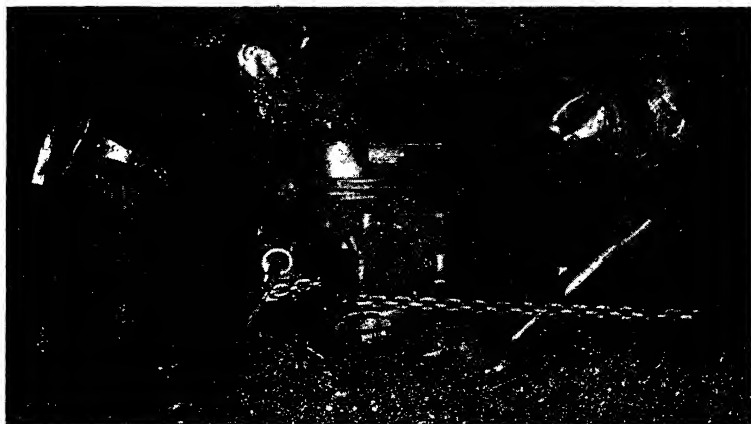


FIG. 2.—UNDERCUTTING COAL DRY.

Discussion of Data

It will be noted that there was a wide variation in these figures in different mines, ranging from 26,541 to 1055 million particles weighing from 752.8 to 61.8 mg. per c. m. More significant than the actual size of the figure is the fact that in each mine where averages were arranged by operations in order of magnitude, the figures for undercutting were greatest, excepting only dust raised by mechanical drilling. As drilling affects much less material than undercutting, the total amount of dust raised by undercutting is considerably greater than that produced by drilling.

The chief factors that entered into this wide range of results in different mines were: Character of coal, experience of operatives, condition of equipment, moisture conditions in working places (all of which affect the total dust produced), and circulation of air at or near working

faces (which controls the dilution of the dust). Friable coal, inexperienced operatives, dull bits on mining machines and dry working places tend to increase dust; while hard coal, experienced machine men, sharp bits on machines and damp working places tend to reduce dust. Poor circulation of air at the face means an increase in the amount of dust in each unit of volume of air; good circulation of air in working place dilutes and removes the dust as it is produced, and so lowers the figures.

Dust Produced by Hand Drilling

Table 4 gives details of dustiness in mines using hand auger drills. The drilling was in shale roof $3\frac{1}{2}$ to 5 ft. above the floor, and in coal face about 2 ft. from the floor.

TABLE 4.—*Dustiness by Mines When Drilling with Hand Auger*

Mine	Number of Samples	Positions of Holes ^a	Millions of Dust Particles per Cubic Meter of Air			Milligrams of Dust Particles per Cubic Meter of Air		
			+10	-10	Total	+10	-10	Total
<i>F</i>	1	80°	75	3937	4012	68.3	40.7	109.0
<i>K</i>	1	90°	12	2961	2973	30.6	15.4	46.0
<i>L</i>	2	15°	15	1404	1419	22.4	8.5	30.9
<i>O</i>	1	0°	8	89	97	5.7	8.3	14.0

^a Position with reference to the horizontal.

Discussion of Data

These results show a comparatively high range in amount of dust encountered but the least dust of any of the drilling operations studied. In the three mines showing the largest amount of dust, drilling was in shale roof which afforded opportunity for the dust to scatter and be distributed by the ventilating current in falling. In two of these mines drilling was being done nearly vertically. These samples show the maximum dust conditions. The smallest amount of dust was raised while drilling horizontally in coal at the face.

Hand auger drilling is a decidedly slower process than mechanical drilling; consequently there is better opportunity for air currents to remove the dust while it falls from the hole to the floor. This undoubtedly is why the operation produces much less dust than either of the mechanical methods. The variation between individual samples and mines can be easily explained by the kind of material drilled, the angle and depth of drilling, distance through which cuttings fall, condition of drill, continuity of operation; and air movement at the point of sampling.

Dust Produced When Loading Coal

Table 5 shows the results of sampling when coal was being loaded:

TABLE 5.—*Dustiness by Mines When Loading Coal*

Mine	Number of Samples	Millions of Dust Particles per Cubic Meter of Air			Milligrams of Dust Particles per Cubic Meter of Air		
		+10	-10	Total	+10	-10	Total
<i>M</i>	10	47	2758	2805	107.7	52.7	160.4
<i>K</i>	10	63	2032	2095	96.4	37.2	133.6
<i>D</i>	2	67	1373	1440	93.6	41.1	134.7
<i>F</i>	7	25	1130	1155	35.6	12.8	48.4
<i>O</i>	3	66	406	471	51.8	24.0	75.8
<i>H</i>	2	16	455	471	18.0	0.6	18.6
<i>L</i>	6	7	224	231	10.0	0.8	10.8

Discussion of Data

It will be noted that there was a wide range of dustiness during loading operations, varying from 2805 to 231 million dust particles per c. m. of air and from 160.4 to 10.8 mg. The size of coal loaded ranged anywhere from large lumps to machine cuttings; the number of loaders from 1 to 3, averaging two loaders per car; and the number of cars loaded from 1 to 4, averaging about one or two cars per sample collected. In all mines, with the possible exception of the mine in which pick work was employed, the size of coal, generally speaking, was ordinary run-of-mine. Where pick mining was used, large lumps, half the size of a man in many instances, were cut out of the solid and lifted into the car. In this mine, minimum dust conditions during loading were encountered. The worst condition in loading was found while "bug dust" was being loaded. One such sample showed 10,134 million particles, weighing 677.3 mg., while the average of 5 similar samples was 3510 million particles, weighing 246.1 mg. per c. m. of air, amounts to more than twice the average of the loading operation as a whole. If water had been used in cutting, much if not most of this dust in the air would have been avoided.

The principal factors affecting the concentration of dust raised during loading can be summarized as follows: size of coal, number of coal loaders, number of cars loaded, character of coal loaded, method of winning coal at face, moisture conditions, air movement at working face, activity of the men, and number of times coal is handled. Fine coal, of a friable nature, tends to form more dust than harder material, while the more men working, the more cars loaded, the faster the work, the more times the coal is handled between face and car, the more dust will be stirred up.

Dry coal will tend to increase the amount of dust in the air and lack of ventilation will allow dust to accumulate in the surrounding air.

Dust Produced When Undercutting Wet

Table 6 shows the dust produced when water is sprayed on the cutter-bars (Fig. 3) of machines. Water at the rate of 30 to 50 gal. per 100 sq.

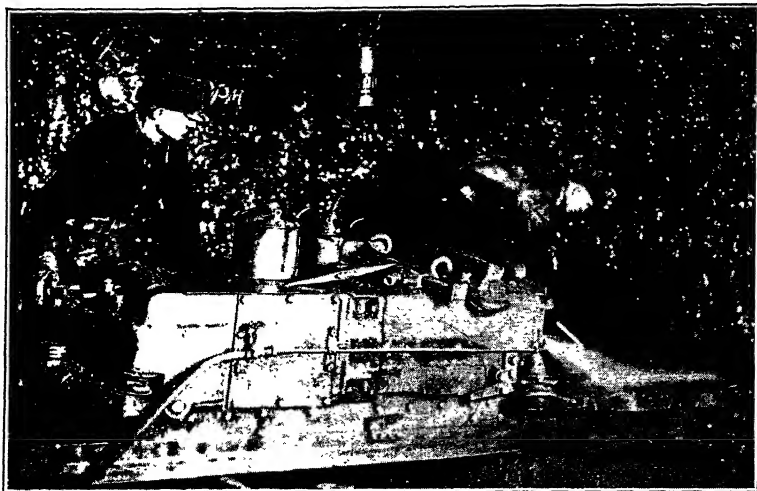


FIG. 3.—UNDERCUTTING COAL WITH WATER ON THE CUTTER BAR.

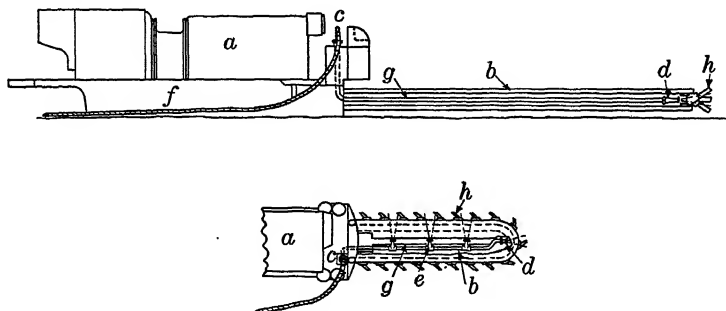


FIG. 4.—METHOD OF APPLYING WATER AT FAR END OF BAR.

a, Mining machine; b, cutter bar; c, connection of flexible hose to machine; d, end nozzle for water discharge; e, side nozzles; f, flexible water hose; g, water pipe to end of bar; h, bits.

ft. of undercutting was applied under pressure through $\frac{1}{4}$ to $\frac{1}{2}$ -in. pipes either on the ingoing bits near the machine, or sprayed over the cutter-bar near the machine on the ingoing side, or at the far end of the cutter-bar. (Figs. 4, 5 and 6.)

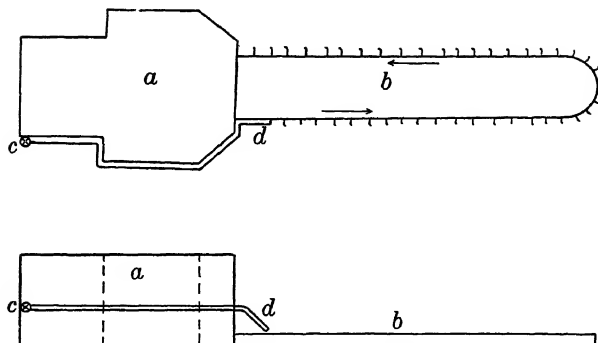


FIG. 5.—METHOD OF APPLYING WATER ON INGOING BITS CLOSE TO MACHINE.

a, Mining machine; *b*, cutter bar; *c*, connections of flexible hose to machine; *d*, point of water discharge.

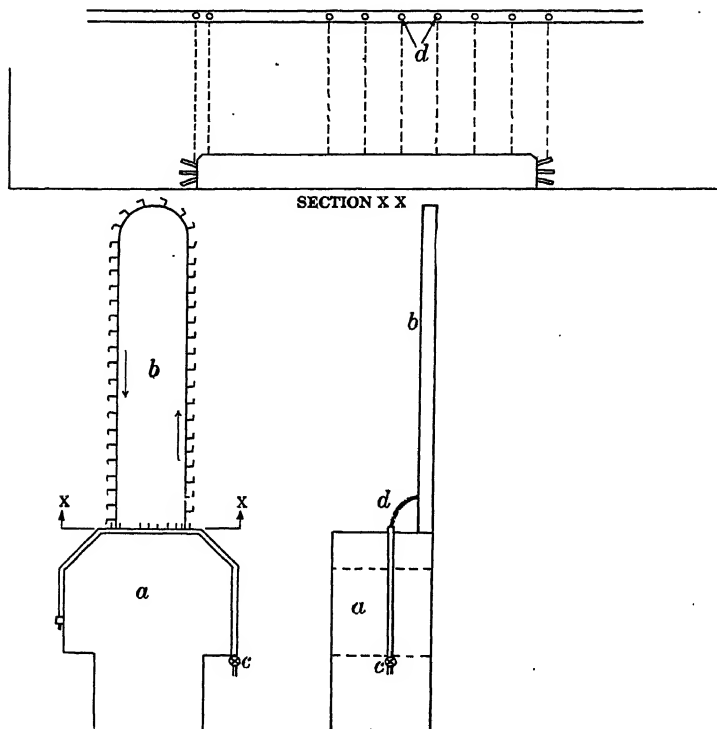


FIG. 6.—METHOD OF APPLYING WATER OVER HALF SECTION OF CUTTER BAR ON INGOING BITS.

a, Mining machine; *b*, cutter bar; *c*, connections of flexible hose to machine; *d*, points of water discharge.

TABLE 6.—*Dustiness by Mines When Undercutting Wet*^a

Mine	Number of Samples	Millions of Dust Particles per Cubic Meter of Air			Milligrams of Dust Particles per Cubic Meter of Air		
		+10	-10	Total	+10	-10	Total
<i>E</i>	1	61	1337	1398	25.5	27.1	52.6
<i>G</i>	2	22	1382	1404	24.0	25.3	49.3

^a Compare with Table 3, undercutting dry.*Discussion of Data*

Although only a few samples are represented here, their close agreement would seem to point to their reliability. It is very apparent that the use of water on the cutter-bar greatly reduces the amount of dust raised. Two mines sampled under these conditions showed averages of 1398 and 1404 million dust particles weighing 52.6 and 49.3 mg., respectively, raised into each cubic meter of air breathed by the men. A certain amount of dust was raised by the movement of the mining machine and men in the working place before cutting was commenced, and also by the picking in roof necessary to set the jack pipe.

Dust Produced When Pick Mining and Loading Coal

Table 7 gives the amount of dust raised when pick mining and loading coal was under way:

TABLE 7.—*Dustiness by Mines When Pick Mining and Loading Coal*

Mine	Number of Samples	Millions of Dust Particles per Cubic Meter of Air			Milligrams of Dust Particles per Cubic Meter of Air		
		+10	-10	Total	+10	-10	Total
<i>D</i>	1	31	1728	1759	11.3	14.1	25.4
<i>K</i>	2	24	731	755	19.1	15.5	34.6
<i>O</i>	7	49	423	472	27.7	14.7	42.4

Discussion of Data

In the three mines in which these samples were collected the range in dustiness was from 1759 to 472 million particles per c. m. of air, and from 25.4 to 42.4 mg., respectively. Pick mining was practiced chiefly where it was necessary for the miner to pick down enough coal to complete loading a car. It raised less dust than loading in the absence of picking. This was due to the intermittent loading, which gave the dust opportunity to clear while picking.

TABLE 8.—*Dustiness by Mines during Haulage Operations*

Mine	Number of Samples	Millions of Dust Particles per Cubic Meter of Air			Milligrams of Dust Particles per Cubic Meter of Air		
		+10	-10	Total	+10	-10	Total
Mules							
<i>M</i>	1	30	1829	1859	44.4	17.0	61.4
<i>P</i>	8 ^a	5	598	603	4.4	3.1	7.5
<i>B</i>	2	8	44	44	0.4	0.5	0.9
Motors							
<i>L</i>	3	8	504	512	0.0	0.2	0.2
<i>M</i>	3	3	331	334	16.1	9.2	25.3
<i>K</i>	4	6	297	303	8.5	31.6	40.1
<i>H</i>	3	9	227	236	10.9	5.4	16.3
<i>F</i>	7	8	172	180	3.2	1.4	4.6
<i>E</i>	4	2	145	147	0.0	0.0	0.0
<i>O</i>	2	16	86	102	0.0	0.0	0.0
<i>N</i>	8	1	79	80	0.4	0.2	0.6
Ropes							
<i>P</i>	9 ^a	3	231	234	3.4	1.5	4.9
<i>N</i>	8	2	86	88	1.3	0.9	2.2
<i>I</i>	6	0	39	39	0.6	0.2	0.8

* Samples collected in entries which had been rock-dusted 6 months previously, but which were ready for re-dusting.

Dust Produced by Haulage Operations

Table 8 shows the concentration of dust raised by mules, motors and rope haulage. This dust is due to the sweeping of fine particles from tops of loaded cars, the grinding of fallen coal by wheels of many trips, and the seepage of fine particles from cracks of cars onto roadways and, to a lesser extent, to the agitation of the dust deposited on ribs, roof and floor.

Mule haulage stirred up the most dust. It is evident that mules, in traveling back and forth from working places along haulageways, raise decidedly more dust than motors or ropes, due to kicking and scuffing by the animals of material between the rails. Furthermore, the constant trampling grinds this dust to a finer size which, when raised in the air, tends to remain suspended unless removed by ventilating currents. This fact is brought out by comparing the dust made by mule haulage and by motor haulage and rope haulage.

The maximum dust condition was shown in a mine represented by a single sample with a total count of 1859 million particles per cubic meter of air. In another mine, eight samples were taken which averaged 603

million particles. The minimum dust conditions were found in a third mine where water was used plentifully at the face while coal was being undercut, after coal was broken down and where loaded cars were sprayed before leaving working places. Here two samples taken in development entries showed a total count of 44 million particles per cubic meter of air. This striking comparison between samples collected where dry methods were used exclusively at the face as against wet methods, shows how the latter affect the dust raised along the haulageways.

Under motor haulage in Table 8 are shown a large number of samples taken in eight different mines. The count of dust particles per cubic meter of air ranged from 512 to 80 million. The minimum average (80) was obtained from eight samples collected in a mine where water was used liberally, not only spraying incoming and outgoing trips of loaded and empty cars, but using water on the cutter-bar and drenching coal after being broken down.

Under rope haulage are shown a number of samples taken in three different mines in widely scattered states. It will be seen that the count ranged from 234 to 39 million particles per c. m. of air. Here again the low count of 39 million is an average from a mine where water was used plentifully at the face, and where incoming and outgoing trips were watered.

The weight of dust in mg. per c. m. of air parallels very closely the respective counts.

PREVENTION OF DUST BY THE USE OF WATER

The principal factors that enter into the prevention of coal dust as determined by this study are the application of water to the cutter-bar of mining machines, watering coal before and after loading, water sprays at main partings or key points, and spraying ingoing empty cars.

The most prolific source of coal dust as brought out by this study is dry undercutting in coal mines. It will be observed by reference to Table 1 that dust from undercutting dry averaged 9810 million particles per c.m. of air, but where water was applied to the cutter-bar, the average was 1402 million, or approximately one-seventh that when undercutting dry.

In addition to the use of water on the cutter-bar, water was applied in a number of instances by miners with a hose to the coal after it had been broken down. They also washed the face of workings and drenched the loaded cars before these left the working places. That less dust was found in the air of haulageways in those mines in which water was used plentifully at the face than in those using dry methods is shown in Table 8. It is evident that in such a case less dust is present in the air of the working places to be carried by ventilating currents, also, less dust can be swept from cars which have previously been drenched with water.

Furthermore, the dripping of water from such cars will keep the road dust damp, thus making it difficult for moving air currents to sweep it into the atmosphere.

In addition to these factors, the use of water sprays at main partings or other key points underground to wet loaded or empty cars, which have lost some of their moisture, is another means of reducing the atmospheric dustiness. These losses may occur during delays and by exposure to moving air currents occasioned by long hauls.

The spraying of ingoing empty cars provides another means by which to reduce the dust in the air. A certain amount of coal dust remains in the cars after they have been dumped. The spraying of these cars before entering the mine to a large extent prevents this dust drying if damp, or falling from or through the car if dry. This was practiced to some degree in some of the mines where this study was made and no doubt contributed to the low concentration of dust found in the air of certain haulageways.

Mechanical drilling has been shown to raise much more dust than any of the other mining operations. In some instances, this is not coal dust, but rather the sandstone or shale material that overlies the coal. However, there is a definite hazard connected therewith which would be greatly reduced by the substitution of wet for dry drilling methods. It has been repeatedly shown by investigations in metal mines that wet drills raise but a fraction of the amount of dust produced by corresponding dry tools. Coal as well as metal mines should avail themselves of this improved practice.

OTHER DUST-PRODUCING AGENCIES AND THEIR REMEDIES

There are certain other agencies which contribute to the amount of dust in a mine, but on which no figures have here been given. Inasmuch as they can easily be remedied and a certain amount of atmospheric dust therefore avoided, it has been thought wise to include them.

Coal dust spilled² along haulageways can be caught and raised into the air by air currents caused either by ventilation or by the movement of trips. A great deal of such spillage can be prevented by the use of tight end cars in place of the loose end-gate type so much in vogue.

Lump coal which falls from trips onto the tracks is soon broken and ground into dust by the passage of trips, as shown in Table 8. This can then be stirred up by all air currents which pass. A minimum topping of all cars would reduce this spillage to a negligible quantity.

² H. P. Greenwald: *Explosibility of Coal Dust from Four Mines in Utah*. Bureau of Mines *Tech. Paper* 386, (1927). Experiments on Utah coals show very definitely that spilled pulverized coal dust between and immediately outside of track rails may propagate an explosion even where the ribs, roof, and floor near the ribs have been rock-dusted.

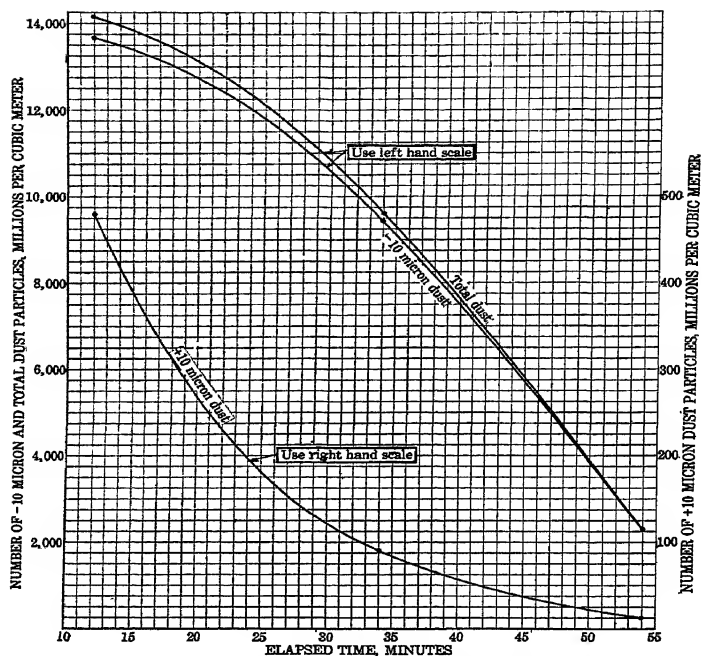


FIG. 7.—DIFFERENTIAL SETTLING OF ATMOSPHERIC COAL DUST.

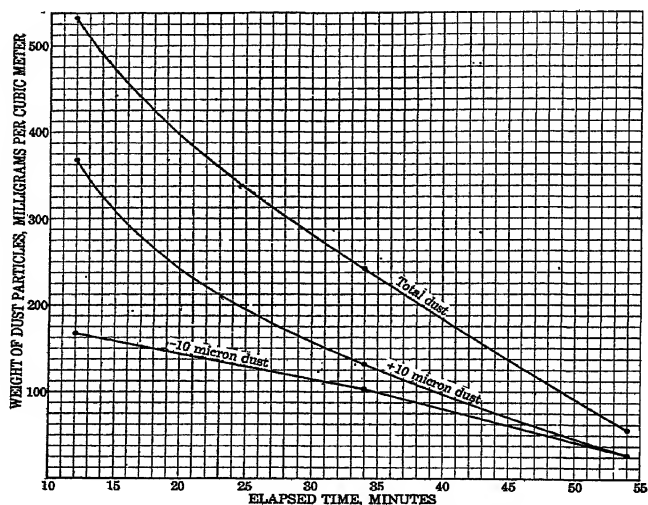


FIG. 8.—DIFFERENTIAL SETTLING OF ATMOSPHERIC COAL DUST.

Many mine cars are allowed to remain in very poor repair. Often wheels have worn unevenly or are broken so that they are not true. Rapid movement of such a car would tend to shake coal off the top and through the cracks.

Much dust on haulage roads is due to uneven, poorly laid track. For this reason all track should be carefully laid and all installations to

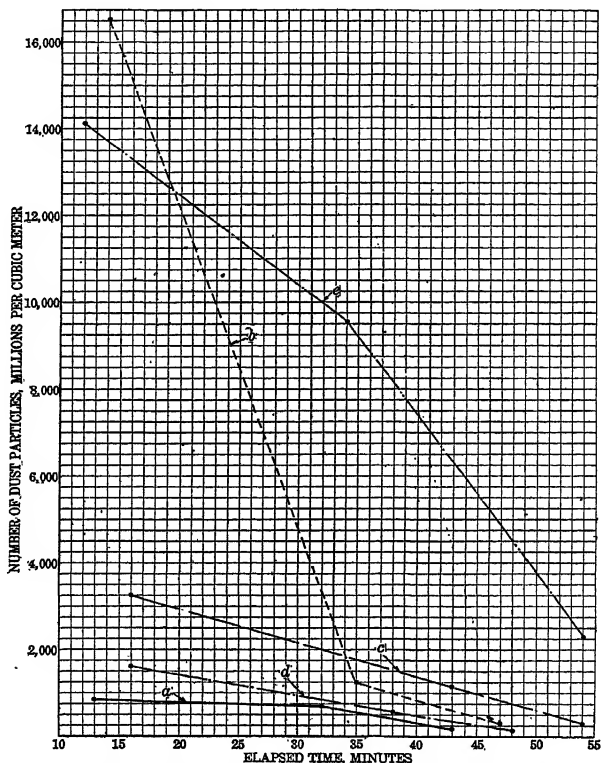


FIG. 9.—SETTLING OF ATMOSPHERIC COAL DUST.

be used over a long period should be well graded, ballasted with rock, and well maintained. The ballast should be gravel, surface soil, slag, or cinders, but never coal. Maintenance should include the frequent cleaning of track by company men definitely assigned to this job.

REMOVAL OF DUST

Although the liberal use of water greatly reduces the amount of dust formed, it should be supplemented by sufficient quantities of circulating air to remove any dust that may be raised. In nearly all of the samples

taken at the face there was little or no movement of air and as a result some extremely fine dust remained suspended over a considerable period. This condition could have been obviated by supplying moving air at the face of workings.

Figs. 7 and 8 show graphically that dust below 10 microns in size remains suspended for a longer period than dust above 10 microns. This smaller size of dust is probably more harmful than the larger particles because of the large amount of surface exposed, hence it should be removed by ventilation as quickly as possible.

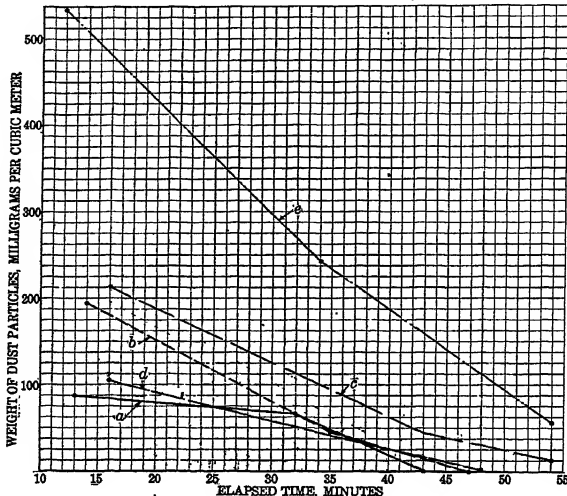


FIG. 10.—SETTLING OF ATMOSPHERIC COAL DUST.

The free settlement of dust where there was almost no air movement is shown graphically for five separate samples in Figs. 9 and 10; in Fig. 9 the total counts are plotted, while in Fig. 10 the corresponding weights are used and lettered similarly. These graphs show that if sufficient time is allowed, dust will settle and so be removed from the air, but the quickest and most efficient method for its removal is by providing sufficient circulating air to the face of workings to remove it as formed.

ADVANTAGES IN DECREASING THE AMOUNT OF COAL DUST

Because the presence of coal dust in mine air presents an explosion hazard, the question of decreasing its amount is extremely important. The Bureau of Mines has recommended that all coal mines be rock-dusted as an explosion preventive. This investigation forcibly brings out the fact that the liberal use of water at the face of workings greatly reduces the amount of coal dust raised. It is not the intention of the writers to

advocate the use of water in place of rock-dusting back from the face of workings, but rather to supplement it. Furthermore, it would be an economical measure to use water at such points because, if less coal dust was raised into the air and carried by ventilating currents to be deposited on the ribs, roof and floor over the rock dust, the life of an application of rock dust would be greatly lengthened.

CONCLUSIONS

The following conclusions are arrived at from this investigation:

1. Jackhammer drilling produced the greatest concentrations of dust of any of the mine operations, 10 times as much as any of the others, and the other dust-producing operations arranged in decreasing order of concentration were electric drilling, undercutting coal by dry methods, auger drilling by hand, loading coal, undercutting coal with the application of water on cutter-bar of mining machines, simultaneous pick mining and loading coal, loading rock, haulage and finally, pick mining.

2. The largest total amount of dust was raised by dry undercutting.

3. Undercutting coal without the use of water on the cutter-bar raised the greatest concentration of dust of any of the common mine operations; loading coal ranked next and haulage last.

4. The chief method for the prevention of dust is the liberal use of water at the face of workings to prevent the dust at its source. It was found that after water had been applied to cutter-bars the amount of dust was reduced to one-seventh of that raised by dry cutting. Moreover, samples taken out by the face of workings where water was used plentifully at the face show a decided decrease in the amount of dust over mines exclusively using dry methods.

5. In almost every instance where sampling was done at the face there was no air movement and the dust was removed almost entirely by free settlement. The best method for removing suspended dust is to provide sufficient circulating air to the face of workings.

6. The ideal combination for explosion prevention is the widespread use of rock dust supplemented with liberal use of water at the face. This elimination of the coal dust at its source lengthens the life of the rock-dust application and necessarily reduces the rock-dust renewals.

DISCUSSION

R. N. HOSLER, Harrisburg, Pa.—Did you make any tests on top-cutting machines?

D. HARRINGTON, Washington, D. C.—I do not think there are any data in connection with top-cutting machines. The use of water on the cutter bar of top-cutting machines is just as practicable as it is on bottom-cutting machines.

R. N. HOSLER.—I think so and perhaps a little more necessary.

D. HARRINGTON.—Very much more necessary. As a matter of fact, it is done very much in Utah. For instance, the Utah regulations force the use of water on mining machines. They use a considerable number of the Jeffrey arc wall and of the Goodman straight-face machines, but they put the water on the cutter bar of the top-cutting machines just as they do with the shortwall machines operated on the floor. I agree with you it is very much more necessary on the top-cutting machines than on the bottom-cutting machines.

R. N. HOSLER.—That is one of the things that the coal mining industry is facing. It is one of the coming machines. In witnessing several of these machines in operation I saw some without the use of water and I saw some with it. I noticed the difference.

Advances in Drilling and Production Methods*

By ROBERT R. BOYD,† BEVERLEY HILLS, CAL.

(New York Meeting, February, 1927)

THE year 1926 has seen few radical changes or developments in methods of drilling and handling oil wells, but it has been a year of steady progress and development in deep drilling and production methods.

Probably the outstanding development of the year was recognition by the directors of the American Petroleum Institute of the importance of natural gas in the production of oil, and the Institute directors' resolutions in December, 1926, regarding regulation of gas production with oil by state legislation. This matter is extremely important and will receive the attention of the industry in 1927.

ADVANCES IN DRILLING EQUIPMENT

The history of the progress of deep well drilling is the solution of problems as they have confronted the operator. As the prospects for deeper production have become apparent, methods of penetrating the earth to greater depths have been devised.

Rotary Equipment

Improvements in the construction and in the methods of operation of rotary drilling equipment have practically caused elimination of cable tools in California for all but shallow work. The rotary drill, from 1910 when it began to be a practical factor in California, has been making steady advances, first displacing cable tools for drilling to the oil sands in proved fields and later displacing the cable-tool outfit entirely. With the perfection of machinery for taking core samples of the formation penetrated, the rotary drill has become generally used for wildcat work, continuous coring being practiced when desirable.

Deep Holes

The use of the rotary drill has made commercially feasible the opening of oil fields at greater depths than was considered possible with cable tools. Heavier equipment has been promptly supplied as occasion has

* For discussion see Petroleum Development and Technology in 1926, p. 30.

† In collaboration with Ward B. Blodget, assistant manager, Chanslor-Canfield Midway Oil Co., Los Angeles, Cal., and N. W. Wickersham, valuation engineer, same company.

demanded it. Successively deeper fields are being drilled and the limit is not yet in sight.

Wells 1 mile deep called for newspaper headlines 4 years ago. Today an entire field (Ventura Avenue) is being developed at depths in excess of 1 mile and has 13 wells flowing, from 6000 ft. or deeper.

In 1925, the world's rotary record was broken when The Miley Oil Co. drilled its Athens No. 6 to a depth of 7591 ft. Casing was successfully landed at that depth and the well was put on production. This is still the world's deepest producing well.

The record as to depth was broken in 1926 by the success of the Chanslor-Canfield Midway Oil Co. who drilled its Olinda 96 to the depth of 8046 ft. and successfully landed a string of 4½-in. pipe in the bottom of the hole. This well is not yet producing but showings of oil and gas were encountered and tests are now under progress for production at depths in excess of 6000 feet.

Use of Electricity

Previous to the latter achievement, the deepest hole in the world has been drilled by cable tools and completed at a depth of 7756 feet.

The development of rotary equipment until recently was along the early established lines. Originally the prime mover was steam. The operators were familiar with steam, which was usually the most practical power available, and, due to excess gas production and cheap fuel, it was considered economical. Because of the capital invested in steam equipment and because of the greater familiarity with its use, steam power was used very extensively in the oil industry long after it had ceased to be the most economical prime mover available.

In line with the needs of the industry, rotary equipment was built larger and stronger. Three-speed draw works were introduced to facilitate handling long strings of pipe. Derricks of 122 ft. became standard, making it possible to handle 4 lengths of drill pipe in the derrick instead of 3 with the shorter derrick. With depths exceeding a mile, more attention was drawn to economies in drilling. The increasing market for gas and the increased value of fuel oil disclosed the fact that the ordinary oil-field methods of using oil and gas to develop steam were exceedingly wasteful.

Electric drilling was first successfully developed in 1922 by the Chanslor-Canfield Midway Oil Co. in the Torrance oil field of California, and its application resulted in a very material power economy, a reduction in labor costs and an increase in drilling speed. The following table shows the saving by the use of electric power on the Del Amo lease of the Chanslor-Canfield Midway Oil Co. in the Torrance field in 1925.

SAVING OBTAINED BY USE OF ELECTRICITY IN AN OIL FIELD

	WELLS DRILLED BY ELECTRIC POWER	WELLS DRILLED BY STEAM POWER
Number of wells.....	12	20
Depth, average, ft.....	3513.5	3620.9
Time elapsed from spudding to completion, average, days.....	42	51.4
Footage, average per day, ft.....	83.7	70.5
Power cost, average per ft., cents.....	12.5	20.5
Labor cost, average per ft., dollars.....	2.66	3.14
Basis of comparison—power cost, 1 c. per kw. hr.; fuel gas cost, 10 c. per M cu. ft.		

Electric power was first applied to a hand-fed rotary outfit. The equipment consisted of a 100 hp. 440-volt, 3-phase, 50-cycle, 10-pole, variable-speed, drilling motor with reduction gears, installed in place of the steam engine. The motor was constructed for plugging and reversing service, and had an ultimate intermittent capacity of 300 hp. The controller was cam type operated with resistances so graduated that the maximum load could be picked up gradually, an essential characteristic. On the mud pumps 50-hp., 440-volt, 8-pole, variable-speed motors were used. Standard pumping equipment consisted of a 15 to 35-hp., 2-speed pumping motor with a suitable reduction unit, which was installed at the time of drilling and was used as an auxiliary in handling drill pipe and casing. With this type of equipment Olinda 96 was drilled.

Danger of Crooked Holes

Town-lot drilling, with the resulting close spacing of holes, forcibly brought to the operators' attention the amount of drift or divergence from perpendicular in rotary holes, as several times holes were drilled together.

Much thought was turned to other losses brought about by crooked holes. The correlations of the formations encountered and the consequent calculations of the depths for setting pipe and finishing wells are based on the assumption that drill holes are vertical. Hence error caused by drill-hole divergence, especially on pitching formations, may be very costly to the operator. Wells that become so crooked in the drilling operations that pipe cannot be successfully landed at the required depth, are eventually redrilled or abandoned. Often because of the continuous wearing of the drill pipe against the upper string of casing, holes are worn through the water string and the well seriously imperiled. A crooked hole once put on production presents a serious operating problem ending usually with an expensive redrilling job, or the ultimate abandonment of the well.

With the serious divergence of drill holes realized and methods of surveying them perfected, the problem of keeping them straight has been approached from more scientific lines. The effect of the different types

of bits on hole divergence is being studied and promises to be an interesting field for investigation. No conclusions have been drawn from data yet assembled.

Method of Surveying Drill Holes

Alexander Anderson has devised an accurate and speedy method of surveying drill holes and has done a great deal of work that is of much value to operators. Opportunities for checking his results have occurred and the accuracy of his work is being established. Anderson's compilation, from the results of the survey of 23 rotary drilled holes, shows a divergence running from 250.6 ft. minimum to 616.1 ft. maximum with an average of 421.6 ft. on a 5000-ft. hole.

Automatic Electric Rotary Drilling

The most signal advance in drilling equipment has been made through the introduction of the Hild automatic differential electric drive, which has now been in use for approximately 3 years. The first outfit was installed on the property of the Chanslor-Canfield Midway Oil Co. at Torrance, Calif., in December, 1923, as an experiment. It was successful from the start and has been in continuous use ever since. It is now drilling its fourteenth hole, has drilled a total of 50,000 ft. with only minor repairs, and is still in good condition. The cost of drilling with the automatic electric rotary shows up favorably with that of the hand-fed electric rotary. It has the additional great advantage of drilling a straighter hole than can be drilled by hand and at a greater speed. The pressure on the bit is automatically varied according to the resistance encountered in the formation, thereby eliminating to a great extent the human element. This gives the driller more freedom and enables him to supervise other work about the rig.

In soft formations maximum drilling speed is maintained, and as the bit enters harder rock causing the drilling motor to slow down, the regulating motor immediately takes up part of the weight of the drill pipe, which removes some of the load from the bit and thus eliminates crowding, the frequent cause of divergence in rotary holes. The outfits are very popular with the drilling crews and are affectionately called "push-button rigs."

Enough holes have now been drilled with the automatic rigs to establish the following advantages: (1) A 50 per cent. reduction in the number of "twist offs" of the drill pipe; (2) a reduction in wear on the drill pipe, tool joints and casing; (3) a decrease in upkeep expense on wells after being placed on pump.

The automatic outfit consists of a differential gear unit with a 75-hp. drilling motor, a 35-hp. regulating motor and necessary control equipment. The motors are capable of developing 435 horsepower.

Improved Draw Works

To take care of this increased power development, larger and more rigid rotary draw works are coming into use. The latest type is built with 7-in. line shafts and 7-in. drum shafts with removable bronze bearings, all six bearings being interchangeable. Three-speed hoists and automatic joint breakers are features of importance in deep work. Make-and-break rotary tables are becoming popular. Steel hoist-drums and semi-steel brake flanges are used in the latest construction; brake flanges are water cooled. Metallurgical improvements are strengthening parts that wear rapidly, increasing their life without increasing their weight.

Improvements in Pipe

Improvements in pipe have been brought about by the adoption of the A. P. I. standard, permitting the use of heavier strings of casing of the same approximate nominal diameter pipe.

Seamless pipe for casing as well as for drill pipe is coming into more general use. The depth to which lapwelded casing may be safely used is being increased due to the greater care being taken by certain manufacturers to meet the needs of the oil industry.

Float plugs are now generally used in landing long strings of casing. In handling drill pipe at great depths, float valves have proved of value.

New Types of Pumps

Mud pumps are continually being made larger and heavier. Dupler power-driven pumps, 7¼ by 14-in., are the latest types used for deep work and to overcome heavy gas pressure. Power pumps with gear drive have been tried, but are not always proving satisfactory. The belt-driven pump is still the most popular as it better meets the conditions imposed by stopped or impeded circulation.

Mud pumps are built with 8-in. suction and 3-in. discharge, capable of sustaining working pressures of 1000 lb. per square inch.

Kinds of Power Used

Where steam power due to economic conditions still continues to prevail, similar advances have been made in rotary draw works, mud pumps and all other accessories.

The use of the Diesel engine as the prime mover has been tried and found successful, very remarkable power economies having been effected with it. The extremely heavy initial investment required for this equipment limits its use to wildcatting in regions where cheap electric power is not available and where fuel is expensive.

The fact that the present record for deep drilling is held by electrically-driven equipment, is largely incidental, and there is nothing inherent in power conditions to prevent the honor at some future time being held by steam or the internal combustion engine.

Advantages of Electric Power

Electric power has great advantages, however, other than that of power economy under local conditions. It has a superior adaptability to the automatic differential drive, which alone entitles it to serious consideration. No other power is capable of such accurate measurement.

The graphic power chart, from the recording wattmeter, used with either the hand-fed or automatic outfit, gives an interesting and extremely valuable record of daily performances. A study of the chart reveals the time and duration of every operation. From the peak loads on pulling out, the driller knows the amount of surplus power available and can tell the time to switch from lower to higher gear. During drilling operations he knows by the amount of power being consumed, the relative traction on the bit and from experience can gage the pressure needed for maximum drilling speed.

As each driller can inspect the charts of the other tours, he can learn the best running load under different conditions, and gain experience from the performance of others. If one driller continually does more work during his tour, a study of the chart will often reveal the cause. A healthy competition is set up among the drillers, each striving to have the best record. The failure of a crew to cooperate with the others is shown on the graphic chart. It has a value to both superintendents and drillers that is not reached by any recording devices for other kinds of power.

By watching his indicating meter the driller knows the amount of power being used and the consequent relative stress being applied to his drill pipe. From a knowledge of the ultimate strength of his pipe he is warned in time usually to prevent "twist offs." If a "twist off" occurs, the graphic meter leaves a record of what transpired at the time.

Anti-friction Tool Joints and Collars

A recent introduction consists of anti-friction tool joints and anti-friction drill-pipe collars. Their success is well established. The tool joints and collars proper are protected by bands turning on ball bearings, which eliminates the friction and wear on the casing and on the drill pipe. The value of this equipment in saving wear on casing has been substantiated in one well in which a string of 8 $\frac{5}{8}$ -in. casing had been cemented at a depth of 4670 ft. At 5800 ft. prolonged fishing and side-tracking work became necessary. Though continuous work has been in progress through the 8 $\frac{5}{8}$ -in. casing for approximately 1 yr., no apparent damage has resulted. Through the use of the ball-bearing joints, the life of the protecting string has been prolonged. The increased life of the tool joints alone has justified the use of the anti-friction type, and the saving by protecting the pipe in the hole and the increase in useful power applied

at the bit make this one of the most promising of the smaller accessories for deep drilling.

Improved Bits

The fishtail bit is widely used; the cutting edges, instead of being dressed in the usual manner, now have special alloys built upon them greatly prolonging the life of a bit. The Hitchcock bit is a modified fishtail bit with removable blades made of special steel alloy, which are replaced when necessary. These bits have been used with success, but cause an additional fishing hazard. Disk bits are much used in fast drilling. Rock bits with cutters of either cone or roller type acquit themselves well in hard formation and are much used in deep holes.

A bit of unique construction invented by J. A. Zublin, is now being introduced. It is offset in construction, and has a tumbling action. Its successful performance depends upon crowding. The natural inference would be that this bit would aggravate the deviation of a hole but, due to its construction, very little gage is lost while drilling, which eliminates most of the reaming necessary with other types of bits. It is possible that this bit will find an extended field of usefulness.

Steel and Dual Derricks

Steel derricks are coming into common use but their economy, except in wildcat work, has not yet been fully established. The dual derrick has been introduced, but apparently its high cost will prohibit its use in California. The convertible feature does not warrant the additional expenditure as the useful life of the movable section is too short.

Use of Mineral Mud

The use of heavy mineral mud as an accessory in drilling wells in areas of heavy gas pressure has been an outstanding advance of the year. Barite was first used successfully, but for commercial reasons hematite has supplanted it. There is now a plant preparing high-grade hematite ore for the purpose of mixing with rotary mud to increase its weight. The average heavy mud used in the Ventura district weighs 85 lb. per cu. ft.; some muds weighing 100 lb. per cu. ft. have been used.

Hematite in the mud has only a slight effect on its viscosity. Consequently the increased weight, which gives additional pressure on the bottom of the hole, does not seriously increase the pump pressure.

ADVANCES IN PRODUCTION METHODS

Deep drilling obviously is resulting in new production problems. It has become necessary to pump wells approximately 5000 ft. deep, although only in a few of these have working barrels been set below 4500 ft. Deep-well pumping requires heavier and stronger equipment.

Stronger Materials Needed for Pumping Equipment

Sucker rods from ordinary good steel that gives satisfactory service in 3000 or 3500-ft. wells, will not operate satisfactorily in wells tubed from 4000 to 5000 ft. When $2\frac{1}{2}$ or 3-in. tubing is used, as is common practice in California, total loads including impulses may stress $\frac{3}{4}$ -in. rods to 40,000 lb. per sq. in. As rod stresses are repetitive, steels of quality to withstand 24 or 26 maximum stresses of 40,000 lb. per sq. in. per min., must have yield points well in excess of this figure. Development of stronger rod steels, as well as full-length heat-treating furnaces for sucker rods, are occupying the attention of rod manufacturers today.

The American Petroleum Institute has standardized sucker-rod joints, tubing and pumps, and with this movement has come improvements in design of certain parts of pumping equipment, and the selection of the best and strongest designs in others. The new A. P. I. sucker-rod joint, for example, combines the best principles of several rods of known merit, and engineers believe that now further improvement in rods must come in the form of stronger materials.

A. P. I. tubing joints, both for plain and upset tubing, are long-collared joints with long threads to give strength and ruggedness. Tubing, unlike casing, is subject to continuous thread wear due to pulling and rerunning. By providing longer threads and collars for tubing and on upset tubing, increasing the length of the upset to provide for rethreading, greater resistance to wear and greater life are insured. With tubing, as with other pumping equipment, A. P. I. standards were chosen to meet the exacting service of deep wells.

Except for continuous improvements in the details of equipment, no revolutionary changes in beam-pumping equipment occurred during the year. Long-stroke pumps, though still in use, have never gained the widespread favor that was expected 2 or 3 years ago. Individual pumping units designed to eliminate friction losses and pump wells with less power are finding a good field in parts of the Mid-Continent and Gulf Coast. The units are particularly adapted to electric pumping where the power costs are directly measurable by the power consumed. Also, they are adaptable to wells that do not require frequent or extensive bailing.

Gas Lifts

The gas-lift method of producing wells has been further developed in 1926 and has given unusually good results in the Garber, Tonkawa, Brame and Seminole fields of Oklahoma. These fields all produce high-gravity oils of relatively low viscosities. Many wells in these fields were large flowing wells when first completed, but declined rapidly. Many of them stopped flowing abruptly, although they were producing as much

as 1000 bbl. per day. Previously in the Mid-Continent fields, such wells have been swabbed for production, which is known to cost approximately \$100 per day in those fields. Now many such wells there are flowed by gas lift for as little as \$12 or \$15 per day. More important, however, these wells yield from 500 to 3000 bbl. of oil per day with the introduction of a surprisingly small quantity of dry gas. Following is the record of a typical gas-lift well in the Tonkawa field:

DATA FOR GAS-LIFT WELL IN TONKAWA OIL FIELD

Daily oil production.....	1100 bbl.
Dry gas circulated.....	350 M. cu. ft. per day
Wet gas returned.....	735 M. cu. ft. per day
Gas-oil ratio, gas from the formation.....	350 cu. ft. per bbl.
Flowing pressure.....	230 lb. per sq. in. (gage)
Direct gas circulating cost (approximate)....	1.1 c. per bbl.

Such results as these are much more favorable than average results in California, which might be illustrated by the following record.

DATA FOR GAS-LIFT WELL IN DOMINGUEZ FIELD

Daily oil production.....	550 bbl.
Dry gas circulated.....	550 M. cu. ft. per day
Wet gas returned, total.....	1100 M. cu. ft. per day
Gas-oil ratio, gas from the formation.....	1000 cu. ft. per bbl.
Flowing pressure.....	140 lb. per sq. in. (gage)
Direct gas circulating cost (approximate)....	3.5 c. per bbl.

Although much experimenting has been done in California, efficiencies that even approach those for the Mid-Continent are seldom obtained. Several reasons may be advanced for this. California flowing wells, excepting in one or two fields such as Richfield, have higher gas-oil ratios than those in Seminole field, for example, where natural flowing gas-oil ratios are often 500 cu. ft. per bbl. The result is that California wells usually flow to productions of 100 or 200 bbl. per day, unless some outside factor such as water causes them to stop. Also, California wells frequently flow 1 or 2 years before reaching the end of their natural flowing lives and remove much more oil from the formations than Mid-Continent wells, before they die.

Although it cannot be said with certainty, it would appear that the nature of the production formations in the Mid-Continent wells, mentioned previously, are conducive to more free flows of oil, and thus would feed more fluid into the hole in a given length of time than the average California well. This point is almost proved by certain wells in northern Oklahoma that have produced over 1000 bbl. per day by gas lift for several months.

Despite its costs, for gas lift is a relatively inefficient pumping method mechanically, it is certain that the gas lift has come to stay. Its most

useful application will doubtless come in future years when wells 6000, 7000 and 8000 ft. deep, or perhaps deeper, must be pumped.

The Marietta Repressuring Process

The Marietta process has become a recognized and common production practice in Oklahoma and Kansas. Throughout the old Bartlesville sand fields of northeastern Oklahoma, compressed gas and air plants for "repressuring" the sands may be seen on every hand. These plants, which are being used to revive the production of old properties that were near abandonment, are usually fitted with automatic controls which make possible the reduction of labor and, therefore, plant expense. Although all these ventures have not proved successful, in the main the Marietta process can be relied on to increase production in these fields, and the results have been good enough to induce large operators to buy up many tracts of very small production, in solid blocks wherever possible, with the expectation that the Marietta process will revive them into profitable producers.

In California the Shell Co. of California and the Union Oil Co. are cooperating to recharge the oil zone in the Dominguez field. This experiment, which has been going on for some 5 months, is of unusual interest, not only because it is the first large-scale experiment of its kind in California, but because conditions in the field are entirely different from conditions in the usual Mid-Continent or Eastern fields where the Marietta process has been used. Dominguez wells are from 4100 to 4300 ft. deep and produce 30° A. P. I. gravity asphalt-base oil from unconsolidated sand. The producing zone is 400 ft. thick and is comprised of several layers of oil sands and sandy shales. Wells are cased through this zone with perforated pipe to prevent caving.

At the time the Marietta process was started in California, pressures in the oil sands at the key wells there were over 400 lb. per sq. in. The surrounding producing wells were all flowing naturally or by gas lift, in contrast to the "pint and gallon" wells of Eastern and Mid-Continent fields where the Marietta process has been so successful.

Results of the Shell and Union experiments are already lending encouragement, the Union Oil Co. having reported several hundred barrels daily increase from 4 wells surrounding the key well. Last reports from the Shell Company indicated that three near-by flowing wells were no longer declining, indicating that the introduction of gas was maintaining their production rates.

GAS CONSERVATION

Doubtless the outstanding development of the year in oil production was the action of the directors of the American Petroleum Institute in endorsing the principles of gas conservation, which should lead to greater

recoveries of oil from sands. This attitude of the industry's leaders represents a great stride in production engineering, for engineers began developing the principles of oil recovery and preaching the doctrine of efficient use of gas in recovering oil only a few years ago.

Gas conservation in oil production is now a recognized principle, and to certain engineers, who are numbered as members of the American Institute of Mining and Metallurgical Engineers, as well as to engineers of the U. S. Bureau of Mines, must go the credit of bringing this important problem to the attention of the industry's leaders.

Although praiseworthy in the extreme, the American Petroleum Institute's action at the recent meeting in Tulsa is fraught with danger. Recommendation of legislation to prohibit gas waste is one matter, but writing a law that will be equitable is another. Conservation laws should be general in character and administered in specific cases by experienced production engineers. Attempts to set a maximum limit on gas-oil ratios in a statute, as has been suggested, would often be unfair and confiscatory both to oil operators and royalty holders.

The greatest danger at present, therefore, lies in haste. This subject is new to the industry and to the state and Federal Governments. Many detailed problems must be considered and solved fairly before laws can be enacted. To expect state legislatures to enact wise laws regarding the details of this extremely technical subject is unfair; to advise them wisely at this time is not within the power of the industry's best informed production men.

FIELD STORAGE AND OIL-HANDLING FACILITIES

Although the industry has been making rapid strides in drilling and production methods, it has also been advancing in a field less frequently discussed, but certainly very important. The production and disposition of crude oil requires a tremendous investment in storage tanks of both large and small capacity, and in oil and gas collection and distribution lines. The use of gas traps, or separators, for conserving natural gas, as well as oil, is becoming a universal practice. Gas is usually collected from traps under vacuum, which is held on the gas-collection system by vacuum pumps at the gasoline plant. Except in new fields where time has not permitted the installation of gasoline plant equipment, natural gas from wells is collected and treated for gasoline, the dry or residue gas being sold to gas companies or used for fuel.

The prevention of evaporation losses from crude oil in the process of handling or storing in the field is now an accepted and important part of production work. Gas-tight tanks with relief valves have been used for several years in the Mid-Continent fields, and certain companies have installed vapor-collection systems for recovering gas from storage tanks and conveying it to gasoline plants for the extraction of gasoline. In

California two of the largest operators are now installing vapor-collection or so-called breather systems on well and on lease tanks.

The floating roof has also been accepted by most oil companies as a practicable means of preventing evaporation loss and protecting against fire, but until 1926 very few floating roofs were used on small tanks such as individual well tanks. One company in California, however, has equipped well tanks with floating roofs, preferring this means of preventing evaporation loss to the vapor-collection systems used by other operators.

SUMMARY

The year has been very satisfactory to the industry. Steady improvement is being made in drilling machinery and drilling methods, and producing wells of depths unheard of a few years ago are now common. In 1927 the depths of producing wells will no doubt be extended from about 6500 to 7500 ft. in the Ventura field. Such wells will bring within reach of the drill, great reserves of oil and will appreciably increase the nation's oil resources.

Production methods are keeping apace with drilling methods, and the industry now generally recognizes the importance of gas conservation and recovery efficiency. With the recognition of this principle, the new year should see still greater strides in the development of production methods designed to recover oil with the smallest possible quantity of gas.

Advances in Refining Technology during 1926*

BY CHARLES H. OSMOND,† NEW YORK, N. Y.

(New York Meeting, February, 1927)

THE rapid progress of basic changes in refining processes, which has characterized this division of the petroleum industry during the last 7 years, slowed up in 1926 and the industry as a whole devoted itself to the enormous task of adopting these changes to its methods of manufacturing. If for no other reason, the year of 1926 will stand out as the first year of widespread activities in placing refining on a true manufacturing basis.

A brief survey of each of the important operations and the trend of the changes with their economic results indicate the importance of this period of transition. No attempt has been made to detail the numerous activities in each of the major operations and the writer would refer those seeking such information to the various trade papers and the paper on "Basic Changes in Refining Processes," by Walter Miller.¹

Distillation.—Distillation (excluding cracking) is the major item in refining. During the past year there has been a wide application of the improvements developed in former years. Operating costs have been reduced for fuel, labor and capital items. Practically all types of distillations are now carried on as semi-continuous or continuous operations, rather than by the old batch method. Heat-saving devices, such as heat exchangers, or improved combustion chambers, have had a marked effect on the total fuel bill of the industry. The modern pipe-still has been widely used and figures available show that approximately 10 per cent. of the older shell equipment was replaced by pipe-stills during 1926. The range of usefulness of this type of equipment has been extended successfully to the most complicated types of lubricating oil manufacture. These stills afford marked economies in fuel as well as labor costs, also many advantages in older plants where space is limited, as a single pipe-still can replace several batteries of shell stills. Capital expenditure per barrel capacity is decidedly lower and offers economy in this direction. Although there has been considerable discussion of other types of distillation equipment such as the vacuum still, there has been no appreciable increase in these installations during 1926.

Cracking.—Cracking equipment increased rapidly during 1926 as evidenced by the increase in output of cracked gasoline from an average

* For discussion see *Petroleum Development and Technology in 1926*, p. 373.

† Consulting petroleum engineer, Crandall & Osmond.

¹ Walter Miller: *Basic Changes in Refining Processes*. *Petroleum Development and Technology in 1925*, 388.

of 189,000 bbl. per day in 1925 to 249,000 bbl. in 1926. Survey of the present type of equipment shows a decided tendency toward larger units. At the end of 1924 the average unit under construction would produce approximately 300 bbl. of gasoline per day; those under consideration at the end of 1926 are planned to produce 1000 to 1500 bbl. of gasoline per day. These figures alone show the effect of study being given this subject by refining engineers. Naturally this increased daily capacity per unit offers decided reductions in labor and capital costs per barrel processed.

During 1926, operation of most of the cracking installations was greatly improved and we are now getting the accumulated effect of the study given this subject during 1924 and 1925. Judging by the results, this field of manufacturing is progressing faster than any of the others and although a good start has been made, much thought and work still remains until the goal is achieved.

During 1926, the usual number of new processes have been announced, but to date none of these stand out as a guide for the future refiner. Practically all cracking equipment installed during 1926 was of the semi-liquid or liquid phase type.

Fractional Condensation.—No item, except cracking, played a more important part in the improvement programs for refineries during 1926, than the discarding of obsolete separating equipment and the installation of modern apparatus. This change was noted chiefly in the Mid-Continent and Texas areas. Based on the number of such installations in these districts, rerunning was reduced at least 75 per cent. during the past year, and, as every refinery engineer knows, this item alone will show a marked reduction in manufacturing costs per barrel. Results do not rest alone on this reduction of cost, but products are more uniform and yields of desired products decidedly larger. Numerous types of equipment have been given consideration, but judging from this trend of installations and results being obtained, the so-called "bubble" tower offers the greatest possibilities in this field.

DECOLORIZATION AND PURIFICATION METHODS

Light Products.—The increased percentage of cracked gasoline during the past year has forced the study of the treating question of such products. The continuous, closed, sulfuric acid method is more firmly entrenched than ever. None of the new methods proposed during the last 5 years, except the fullers earth vapor-phase process, has shown any appreciable increased application during 1926. The constant introduction of crudes containing higher percentages of sulfur as gasoline producers has made this problem acute. It would seem as if this phase of refining had progressed the least, not due to lack of work however, but largely due to the ever-changing conditions forced upon the industry by varying types of raw material.

Lubricating Products.—The fine-clay contact process for decolorization of lubricating products has been rapidly adopted during 1926. This change is based on lower manufacturing cost and greater recovery. Modifications, based on the type of crude, have been made so that it seems as if this method, under any crude condition, shows the lowest manufacturing cost of any of the commercial methods. Capital expenditures for this method of processing are low as compared with older methods.

The use of mechanical means for de-waxing lubricating oils made from paraffin-base oils has increased at a rapid rate; approximately 11 per cent. of all lubricating oil manufactured in the United States during 1926 was produced in this manner.

Minor Improvements.—Everywhere one sees the evidences of minor improvements in general refinery practice. Coordination of equipment, processing, wide introduction of technical operating forces, combatting corrosion in equipment scientifically, all tend towards lower manufacturing cost. From this record of rapid and important advances in manufacturing, evidently the work of the refinery engineer began to show cumulative results in 1926. It clearly demonstrates the importance of the engineer in the future of an industry which is going to depend more and more for its profits on closely calculated operating costs and related savings.

The Importance of Corrosion Problems*

By F. N. SPELLER,† PITTSBURGH, PA.

(New York Meeting, February, 1927)

INTEREST in the corrosion problem seems to continue without abatement and is bringing practical results. At least four important conferences on this subject have been held during the last 5 months. Those interested in steel-frame structures are taking steps to study vulnerable points on account of the failure of some minor parts in certain buildings after 35 years of service. The steam power-plant group, including the railroads, realize the importance of a better knowledge of corrosion, especially under modern high-temperature boiler operations. The chemical and refrigerating industries have already conducted a systematic study of depreciation of metals by corrosion, for which they have spent considerable money and time during the last 2 years with encouraging results. More has probably been done to control corrosion in the atmosphere and in closed water systems, because the work on these phases of the problem started over 15 years ago and the important factors are now fairly well understood.

As a result of all this work, there is now little excuse for corrosion in water conduits, recirculating condensers and cooling systems, since economical means of prevention have been worked out to meet nearly all situations where such corrosion occurs. The entire Baltimore water supply was treated with lime at a cost of only 35 c. per million gallons and the rate of attack on the pipes has been reduced to not over 20 per cent. of what it was before the treatment. Yearly damage to water mains in Baltimore is estimated at \$300,000, and in New York City at \$5,000,000, not including damage to water conduits in buildings. In some cities corrosion is now recognized by water works' engineers as a major problem.

In the oil industry, there are evidently three general classes of corrosion, relating to: (1) Production, (2) transportation, and (3) refining.

(1) *Production*.—Some fields, those of California for example, have little corrosion trouble in the wells. In others this is a major problem and should be a strong incentive to the development of more resistant metals at a reasonable cost. E. G. Woodruff,¹ consulting geologist, has pointed out that more of the oil of the future will probably come from the

* For discussion see *Petroleum Development and Technology* in 1926, p. 468.

† Metallurgical engineer, National Tube Co.

¹ E. G. Woodruff: *Geological Report on Sulphur Oil Areas. Proc. Am. Pet. Inst.* (1926).

limestone areas rather than from the sands, and oil from the former seems to be accompanied by a higher sulfur content. The corrosive sulfur compounds cause a high corrosion rate even in the absence of oxygen; the sulfur and sulfide of iron deposited upon the metal surface have a strong electrolytic and depolarizing influence, which accelerates the rate of solution of the metal.² The high chromium-iron alloys seem best suited to meet this condition but are at present too expensive for general use. Corrosion from such causes will probably become more serious as time goes on. In the producing departments in certain limited areas, corrosion trouble is now very serious and threatens to limit the operating life of some fields unless an economical remedy is found. Wells costing \$40,000 to drill have been lost through corrosion and the production in others has been considerably curtailed.

(2) *Transportation*.—The actual loss to the industry from corrosion at present cannot be estimated accurately. The chief engineer of the Prairie Pipe Line System recently stated that corrosion in underground pipe lines exacts an annual toll in the United States of at least \$100,000,000. One company in the west reports an annual expenditure of about \$600,000 in combating pipe-line deterioration.

(3) *Refining*.—In oil refineries some progress has been made towards minimizing corrosion of equipment by chemical treatment of the product or by the use of more resistant materials, but corrosion is still a major problem, as is indicated in the recent paper by H. F. Perkins.³

THE TULSA CONFERENCE

The Tulsa conference on corrosion at the annual meeting of the American Petroleum Institute last December was the first general symposium on oil country corrosion to be held in this country. The papers and discussions were divided into three sections including: transportation, refining and producing problems. The meetings were characterized by an intense manifestation of interest, particularly in the subject of protection of transportation lines. The attendance at this meeting was over 300.

The papers on oil line protection brought out the fact that practically nothing had been done to explain soil corrosion or to correlate the physical and chemical characteristics of certain soils with the observed rate of corrosion and pitting in these soils, and further that much more information was required on protective coatings. One author went to the extreme by stating that nothing is known on the subject of protection, but as a matter of fact many lines would not be operating in corrosive soils today if some fairly satisfactory protection had not been found.

² R. Stumper: Researches on the Corrosion of the Cast-Iron Conduit Pipes of the Mondorf Baths. *Revue Technique Luxembourgeoise* (No. 6, June, 1923); also, R. L. Ginter: Report on Chemical Phase of Steel Tank Corrosion Problem. *Proc. Am. Pet. Inst.* (1926).

³ H. F. Perkins: Corrosion in an Oil Refinery. *Petroleum Development and Technology in 1926*, 560.

The ideal protective coating, it was pointed out, must be absolutely waterproof, must adhere tenaciously to the pipe but not to the soil, and must not be injured by expansion and contraction of the pipe. The cost of the coating, life of the line with and without protection, original cost of the line unprotected, and cost of repairs and replacement are important considerations in determining how much can be profitably spent in protecting a line. Some economic calculations including these factors were presented by C. R. Weidner, Prairie Pipe Line Co., who stated that it costs at least \$100,000,000 per year to maintain the underground pipe lines of the United States. As it should be fairly easy to arrive at an estimate of this kind from the records of the operating companies, it is apparent that the magnitude and importance of the corrosion problem in general has probably not been overestimated. The underground gas and oil lines in this country total about 180,000 miles or about three-fourths of the railroad mileage. Fortunately, much of this pipe is laid in soil where there is no serious corrosion.

As a result of the Tulsa conference, a resolution was passed recommending that the directors of the American Petroleum Institute appoint a general committee on corrosion to investigate this subject and do such research work as they deem necessary. This committee has been appointed.

NEED FOR COOPERATIVE RESEARCH

Because of the general situation in the oil and gas industries, an organized and concentrated effort is needed to obtain the requisite data for remedial measures. It seems desirable to centralize research work on this subject in each industry or in groups of similar industries. For instance, the oil and gas groups might very advantageously combine their work in the study of protective coatings and of other methods for prolonging the life of underground pipe. So much remains to be done that any unnecessary overlapping of effort should be avoided. It is not intended to discourage research by individual companies, who on the contrary might well increase their efforts, but to suggest a close cooperation with a central committee representing the industry.

If the study of corrosion is so organized by the American Petroleum Institute, would it not be well for the Petroleum Division of the American Institute of Mining and Metallurgical Engineers to offer its cooperation and obtain representation in this important work? The writer has previously suggested a Central Institute of Corrosion Research, supported by the national government and private interests, with facilities to study the common basic factors of the problem. Whatever is done, however, it is useful to take an inventory of progress periodically. The program arranged for this symposium promises to be another important contribution to the literature on the subject.

The Non-corrosive Ferrous Alloys*

BY JOHN A. MATHEWS,† PH.D., SC.D., NEW YORK, N. Y.

(New York Meeting, February, 1927)

It is no longer necessary to explain to an audience like this that there are stainless or non-corrosive steels. It is still necessary to repeat, and to keep on repeating, that no one of them is stainless or non-corrosive under any and all conditions. We do not make railroad axles, springs and rails from the same kinds of steel; nor are shafts, steering-arms, gears and cap-screws made from the same type analysis. By analogy with other fields of endeavor, therefore, it is logical to expect that all the myriad conditions of corrosion due to inorganic and organic acids and salts and their mixtures in all degrees of concentration and at a wide variety of temperatures and pressures will require different materials to withstand them.

Possibly stainless steel when first introduced a decade ago was expected to do too much, or suffered from overenthusiasm and lack of positive knowledge and experience on the part of its producers. Its name may have been a bit misleading too, for remarkable as this product is, it has its limitations. There are now so many different ferrous products available and calling for recognition, that we may be justified in an attempt at classification. The first division is very simple.

1. Non-corrosive steels that harden by quenching.

2. Non-corrosive steels which do not harden by quenching.

We might make the classification still simpler by saying (1) those that harden by quenching, and (2) those that soften by quenching. Of course there is a border line or fringe of compositions which may possess some of both qualities under limited conditions of treatment. I have seen a steel so perfectly balanced on this border line that in its natural condition, as rolled, the outside of a 1-in. bar was soft and the inside hard, and there was quite a sharp line of demarcation in between. The hardness at the center was more than twice as great as at the surface, as measured by the Rockwell test. This border line of products does not include, however, the commercial products now available to solve the problems of industry.

Let us examine the products of these two classes:

HARDENABLE NON-CORROSIVE STEELS

This group, practically speaking, includes only the products known as "stainless iron and stainless steel." These names have a special sig-

* For discussion see Petroleum Development and Technology in 1926, p. 493.

† Vice-president, Crucible Steel Co. of America.

nificance and should be limited in their use to the straight chromium alloys, as developed by Brearley and Haynes. They are patented and the patents are owned by the American Stainless Steel Co., and a dozen or more of the leading steel companies manufacture them under license agreement. The use of the terms "iron" and "steel" to designate two leading types is quite arbitrary; nevertheless, this usage has now acquired a trade significance that it is well to recognize and thus avoid confusion. Needless to say, both of them are "steels," but the term stainless iron is limited to a product 0.12 per cent. carbon or lower, and usually containing from 12.0 to 14.0 per cent. chromium. This is the commercial stainless iron, even though it is sometimes sold under identifying brand names.

As to the specific field of usefulness of stainless iron, it may be noted that there are many engineering requirements where it is desirable to couple non-corrosive features with good mechanical properties. This alloy, notwithstanding its very low carbon content, responds very well to heat treatment, and furthermore, after hardening it retains its high physical properties up to relatively high tempering temperatures. This gives it valuable properties for use up to moderately high operating temperatures, such as are met in steam-boiler, valve or turbine applications. After quenching a 0.10 carbon stainless iron from 1750° F., it may then be tempered up to 950° F. and not fall below: yield point, 135,000; ultimate strength, 160,000; elongation in 2 in., 17.5 per cent.; reduction in area, 55 per cent.; and Brinell, 325 per cent.

These are very good properties for engineering requirements in a material that is practically unaffected by atmospheric or salt water corrosion, or by nitric acid, alkalis and many other chemicals. By further tempering its corrosion resistance is affected but slightly, if at all, while it acquires greatly increased toughness and resistance to shock with only a moderate sacrifice in elastic properties. For many uses stainless iron does not require heat treatment at all, but to take advantage of its desirable physical properties heat treatment is recommended. It should at least be normalized and it should always be used if possible with a ground, smooth machined or polished surface. The better the surface, the better its resistance to attack. Cooled in air from 1750° F. we may expect approximately the following properties: Yield point, 140,000; ultimate strength, 160,000; elongation in 2 in., 12.0 per cent.; reduction in area, 35.0 per cent.; and Brinell, 300 per cent.

Material such as this has found a large use in turbine blading, valve parts, pump rods, cooking utensils and even for roofing material.

When still greater hardness is needed, as in table cutlery, pocket-knife blades, springs of certain types, and various tools and dies, also for balls, a high-carbon product known as stainless steel is used. It has the same chromium content and usually carries from 0.30 to 0.40 per cent. carbon. This steel differs from stainless iron in that hardening and

polishing are necessary to promote good stainless qualities. In other words, its stainlessness is vastly better when the carbides are in solution; but once in solution it may still be tempered up to 1000° F. without serious decrease in stainlessness. Its initial hardness as quenched may vary considerably by variation in the hardening temperatures between the range of 1650 and 1900° F., and it retains its hardness remarkably well until tempered beyond 950° F., above which it softens very quickly. At a 950° tempering we still have a steel of 200,000 tensile strength with fair ductility. Generally a hardening temperature of from 1750 to 1800° F. is desirable.

With both the iron and the steel, increasing the chromium content above about 14.0 per cent. causes a decrease in hardening capacity but with some gain in stainlessness under severe conditions. Conversely, with chromium constant at about 12.0 per cent., raising the carbon above 0.40 per cent. does little good so far as increasing initial hardness is concerned, and the higher carbons, after tempering at, say, 1000° F., may be softer than the lower carbons for the same degree of tempering and decidedly less stainless. If greater hardness than normal 0.30 per cent. carbon yields is necessary, one must use a steel in which both carbon and chromium are increased.

In both stainless iron and steel we frequently find small additions of silicon, nickel, copper, molybdenum, etc., added for some specific purpose. The normal products are quite strongly magnetic, fairly high in electrical resistance, especially if quenched, and possessing a coefficient of expansion equal or slightly less than ordinary steel. They may be welded with considerable care, as the welding temperature produces a very great grain growth and the material at the weld has a tendency to air harden and to become very brittle. When ability to harden and to obtain high physical properties is desirable, our choice is practically limited to stainless iron and stainless steel.

UNHARDENABLE, NON-CORROSIVE STEELS

Before the advent of the modern steels, containing chromium, we had only the high-nickel steels, usually low in carbon and containing from 25.0 to 35.0 per cent. nickel. These are quite stable so far as atmospheric corrosion is concerned and toward sulfuric and hydrochloric acid conditions are much better than chromium steels but are by no means free from attack. They are, however, many times as resistant as ordinary steels.

As was intimated under the section on hardenable steels, the chromium steels lose this property as the chromium increases, and from about 16 to 30 per cent. chromium with low carbon, we have a series of very resistant alloys, soft and ductile, but weak in shock resistance. They fill a very useful field, when neither great strength nor hardness is necessary

and where great resistance to nitric acid and many chemicals is needed. These alloys also may contain small additions of silicon or copper.

IRON-CHROMIUM-NICKEL ALLOYS •

The most interesting members of the unhardenable alloys are those containing sufficient of both chromium and nickel to become naturally austenitic. E. C. Bain¹ shows just what those bounds of composition are. These steels were studied exhaustively by Dr. Fry and his associates at Krupps. The addition of notable amounts of silicon, usually from 2 to 3 per cent., was found by the investigations of C. M. Johnson of the Crucible Steel Co., to bring about a considerable improvement as regards many conditions of corrosion and resistance to high temperatures and scaling. The good qualities of both the chromium and the nickel alloys are preserved in the austenitic chromium-nickel alloys and the silicon addition still further enhances them, and improves the welding qualities as well as resistance to scaling and some kinds of corrosion. They are so resistant to so many things and conditions that we have given them the name "Rezistal." For the most part they are totally non-magnetic; their electrical resistance is high; to oxygen and hydrogen sulfide at 900° F. for many hours they are perfectly resistant; some of them form no detachable scale in an oxidizing atmosphere above 2000° F. One of the leading chemical firms reports that 25 per cent. nitric acid at 125° F. attacks them at a rate which would require 1000 years to consume a 1-in. plate, while with sulfuric acid of 80 per cent. and upward they might last only one-fourth as long!

Tests might be continued indefinitely but the writer has already discussed at considerable length the peculiar properties of these austenitic alloys in the Henry M. Howe lecture² for 1925, and in an address before the Engineering Foundation³ in 1926. In many respects they are the antitheses of the ordinary carbon or alloy steels. For instance, a coarse grain in them implies ductility and not brittleness, and for this reason their elongation increases and their hardness decreases with ascending quenching temperatures up to 2300 to 2400° F. In sulfurous mine waters they demonstrated superior qualities in the extensive Bureau of Mines⁴ tests of about 50 ferrous and non-ferrous metals and alloys. Many practical service tests are even more impressive. In one coal mine a pump shaft is now running in its fifth year, where formerly 5 weeks was a good run. This shaft cost about 75 c. per lb. but is the cheapest one

¹ E. C. Bain and W. E. Griffiths: *An Introduction to the Iron-chromium-nickel Alloys*. See p. 166.

² John A. Mathews: *Austenite and Austenitic Steels*. *Trans.* (1925) **71**, 568.

³ John A. Mathews: *Present Tendencies in Engineering Materials*. *Mech. Eng.* (1926).

⁴ Coal Mining Investigations. *Bull.* No. 4, Carnegie Inst. Tech. (1922).

they ever owned. In a western mine, water from a precipitation tank totally destroyed a $\frac{5}{8}$ -in. commercial carbon steel plate in 16 days, whereas under the same conditions a Rezistal sample lost 0.20 per cent., in weight. Instances of this kind could be multiplied by examples from many other applications in industry. However, enough has been said and it may be considered that there are now available ferrous alloys of the hardenable or unhardenable types to meet perfectly or almost perfectly every condition of corrosion except that due to hydrochloric acid or aqua regia. Even when not totally non-corrosive they may be several hundred times as resistant as the carbon or medium alloy steels.

The resistance of these alloys to scaling has been referred to. Quite as remarkable as their resistance to corrosion is the display of great strength at the elevated temperatures of the austenitic steels. In this property the chromium-nickel austenitic steels far exceed those of either high chromium or high nickel, while these simpler alloys are far superior to ordinary steels. By way of illustration it may be stated that a 0.25-carbon plate steel will show about 65,000 lb. breaking strength at ordinary temperatures; certain austenitic steels will possess about that breaking strength at 1300° F. or a good red heat. This is all the more remarkable when we reflect that carbon steel suffers a great falling off in strength at about 350° F.; the medium alloy steels at 600 to 700° F. and stainless steel at about 900° F. To put it another way, at 1300° F. certain austenitic steels are two to three times as strong as stainless iron, pure nickel, monel metal and other metals frequently referred to as high-temperature metals, while the strength of carbon steel at 1300° F. is practically negligible.

NEED OF COOPERATION BETWEEN PRODUCER AND CONSUMER

I have endeavored to present a classification and a broad survey of the properties of these newer products of the metallurgists, without burdening you with figures and detail. What I want to emphasize in closing is the need of cooperation between the users and producers. It is particularly necessary that you define and describe your operating conditions with the greatest care. When you try to do so you may find you lack much information about your own processes that it would be valuable for you to have. Sometimes slight changes in conditions totally change reactions. For example, certain metals are quite readily attacked by dilute sulfuric acid, but the action ceases when a very moderate amount of ferric or cupric sulfate is present. Pure aqueous solutions of citric, lactic and acetic acids will attack certain metals that will perfectly resist them in the same strengths when present in lemon juice, sour milk or vinegar, respectively.

It will take the closest cooperation between industry and the steel makers, if we are to avoid costly experiments and mistakes, due to misap-

plications and a lack of complete understanding of the problem. Some of us have been studying these fascinating alloys intensively for years and the more one has studied them the more one realizes what still remains to be done.

The corrosion problems of the giant petroleum industry are not incapable of solution, I am sure, but the more you will tell us about your troubles and the more carefully and accurately you present them, the sooner will solutions be forthcoming.

The steel industry is one of your good customers and the users of our products are also large users of yours. We have a mutual interest in solving your corrosion problems for the ultimate good of those whom we both serve—the general public, and in carrying forward the national program of conservation of natural resources

Review of Production of Petroleum in the United States in 1926*

JAMES H. GARDNER,† TULSA, OKLA.

(New York Meeting, February, 1927)

IT is a striking fact in the domestic oil statistics of 1926 that with an increase in production of 13,000,000 bbl., there was nevertheless a lowering of both crude and gasoline stocks. It was the first reduction in the last eight years. In 1918, the last year of the war, stocks of crude and refined products were lowered 16,000,000 bbl., but in the following seven years stocks were increased 350,000,000 bbl., due largely to the phenomenal crop of major pools discovered in 1923. The big pools of that year coming as they did through some curious coincidence unprecedented in the oil industry, not only piled up a sudden increase of stocks at a time of large imports but they continued to add their quota to an increase of storage until 1926.

With an increase in crude production, according to the *Oil and Gas Journal*, from 760,000,000 bbl. in 1925 to 773,000,000 bbl. in 1926, and an increase in cracking facilities of about 30 per cent., there was a draft on storage of 22,300,000 bbl. of crude and 1,000,000 bbl. of refined. Crude and refined changed from a stock surplus of 29,130,000 bbl. in 1925, to a deficit of 23,300,000 bbl. in 1926, or a differential total of 52,430,000 bbl. With a stock reserve of 541,700,000 bbl. of crude and refined oil in 1925, the amount drawn from storage was not large in percentage but it marked an important contrast with previous years. It meant a year of stability in prices and a sound condition in the industry. The indicated domestic demand increased from 695,120,000 in 1925 to 743,360,000 in 1926, (7 per cent.), 1 per cent. of which was due to California fires. Most of the gain in consumption was due to the increased demand for gasoline. There was little comparative change in the combined imports of crude and refined oil. There was an increase in exports of 17,200,000 bbl. (1.5 per cent.).

A petroleum balance sheet for 1926 and a comparative sheet for the last two years, production in the United States by major districts and pools, and a list of imports are given in Tables 1 to 5, inclusive.

* For discussion see *Petroleum Development and Technology* in 1926, p. 609.

† President, Gardner Petroleum Co.

TABLE 1.—*Petroleum Balance Sheet of the United States for 1926*

(Data from U. S. Bureau of Mines, December estimated. Figures subject to revision.)

	Crude, Barrels	Refined, Barrels	Total Stocks, Barrels
Stocks on Dec. 31, 1925.....	418,400,000 ^a	123,300,000	541,700,000
Production.....	766,900,000		766,900,000
Imports.....	60,660,000	21,500,000	82,160,000
Total oil available during year.....	1,245,960,000		1,390,760,000
Indicated domestic consumption....	834,460,000		743,360,000
Exports.....	15,400,000	113,600,000	129,000,000
Stocks on Dec. 31, 1926.....	396,100,000 ^a	122,300,000	518,400,000
	1,245,960,000		1,390,760,000
Change in stocks during year.....	-22,300,000	-1,000,000	-23,300,000

^a Includes fuel oil in California.TABLE 2.—*Comparative Balance Sheet for 1925 and 1926*

(Data from U. S. Bureau of Mines)

	1925 Barrels	1926* Barrels
Domestic production.....	755,850,000 —	766,900,000
Imports of crude oil.....	62,000,000	60,660,000
Imports of refined oil.....	18,200,000	21,500,000
Total.....	836,050,000	849,060,000
Surplus (+) or deficient supply (—).....	+29,130,000	-23,300,000
Indicated domestic demand.....	695,120,000	743,360,000
Exports of crude oil.....	13,350,000	15,400,000
Exports of refined oil.....	98,450,000	113,600,000
Total.....	806,902,000	872,360,000
Stocks Dec. 31.....	541,700,000	518,400,000

* December estimated. Figures subject to revision.

TABLE 3.—*United States Production by Districts*
[Oil and Gas Journal preliminary figures, 1926]

Fields	1925 Production, Barrels	1926 Production, Barrels
California.....	230,174,000	224,448,000
Oklahoma.....	173,270,000	177,651,000
Texas.....	144,783,000	171,356,000
	1925	1926
Gulf Coast.....	31,192,000	43,252,000
Wortham-Powell Dist.....	34,416,000	26,424,000
Luling.....	9,371,000	7,722,000
Panhandle.....	1,240,000	26,101,000
Reagan.....	8,982,000	11,243,000
Remainder.....		56,614,000
Arkansas.....	78,407,000	58,729,000
Kansas.....	38,151,000	41,346,000
Rocky Mountain District.....	35,390,000	37,713,000
	1925	1926
Wyoming.....	29,229,000	25,465,000
Montana.....	3,863,000	7,744,000
Colorado.....	1,202,000	2,776,000
New Mexico.....	1,096,000	1,695,000
Miscellaneous.....		33,000
Appalachian.....	26,240,000	27,272,000
Louisiana.....	21,537,000	23,282,000
	1925	1926
Gulf Coast.....	3,658,000	4,511,000
Mississippi Valley.....	12,000,000	11,900,000
Total.....	759,952,000	773,697,000

TABLE 4.—*Production from Major Pools Reaching Peak and Others Discovered since Beginning of 1923*

(1926 figures from American Petroleum Institute)

Pools	1923 Barrels	1924 Barrels	1925 Barrels	1926 Barrels
Smackover.....	35,628,000	40,610,000	74,383,000	52,243,000
Long Beach.....	66,768,000	60,220,000	40,173,000	37,756,000
Panhandle.....			1,097,000	26,101,000
Huntington Beach.....	32,430,000	17,548,000	15,877,000	19,303,000
Salt Creek.....	35,302,000	30,832,000	21,584,000	18,055,000
Santa Fe Springs.....	75,248,000	26,450,000	18,970,000	17,654,000
Inglewood.....			18,348,000	17,354,000
Burbank.....	31,699,000	30,291,000	19,955,000	16,500,000
Spindletop.....			380,250	14,689,000
Tonkawa.....	28,033,000	22,209,000	22,722,000	13,866,000
Reagan.....		1,079,000	8,982,000	11,243,000
Garber.....	2,425,000	1,849,000	7,884,000	10,920,000
Ventura Avenue.....	3,711,000	4,050,000	9,263,000	10,813,000
Seminole (incl. Searight)...				10,837,000
Powell.....	31,864,000	33,060,000	17,028,000	10,445,000
Torrance.....	2,765,000	17,528,000	13,242,000	10,275,000
Wewoka.....	1,631,000	3,292,000	4,483,000	9,463,000
Wortham.....			16,838,000	2,834,000
Elk Hills.....	8,180,000	13,686,000	11,971,000	12,227,000
Luling.....	958,000	10,210,000	9,706,000	7,722,000
Dominguez.....		6,813,000	13,329,000	7,754,000
Teeter-Pixley-Scott.....		500,000	4,575,000	6,775,000
Hull.....	7,732,000	7,287,000	7,184,000	6,701,000
Cromwell.....		9,526,000	11,673,000	6,231,000
Rosecrans.....		629,000	7,263,000	6,211,000
Urania.....				4,119,000
Orange.....	5,312,000	4,182,000	5,024,000	3,759,000
Papoose-Gilcrease.....		3,001,000	7,803,000	3,632,000
Haynesville.....	7,365,000	6,723,000	4,739,000	3,579,000
South Liberty.....			4,551,000	2,089,000
Rainbow Bend.....		1,199,000	3,949,000	1,405,000
Totals.....	388,051,000	352,774,000	402,976,250	382,555,000

TABLE 5.—*Imports of Crude and Refined Products into the United States during Eleven Months January–November, 1926, Inclusive*

(From Bureau of Foreign and Domestic Commerce Monthly Reports)

From	Crude Oil, Barrels	Total Crude and Refined Products, Barrels
Mexico.....	38,146,000	53,418,000
Peru.....	2,828,000	2,828,000
Venezuela.....	4,256,000	4,387,000
Dutch East Indies.....	6,457,000	9,721,000
Trinidad and Tobago.....	374,000	619,000
Colombia.....	2,704,000	2,704,000
Ecuador.....	144,000	144,000
Uruguay.....	218,000	218,000
Others.....	268,000	943,000
Total.....	55,395,000	74,982,000

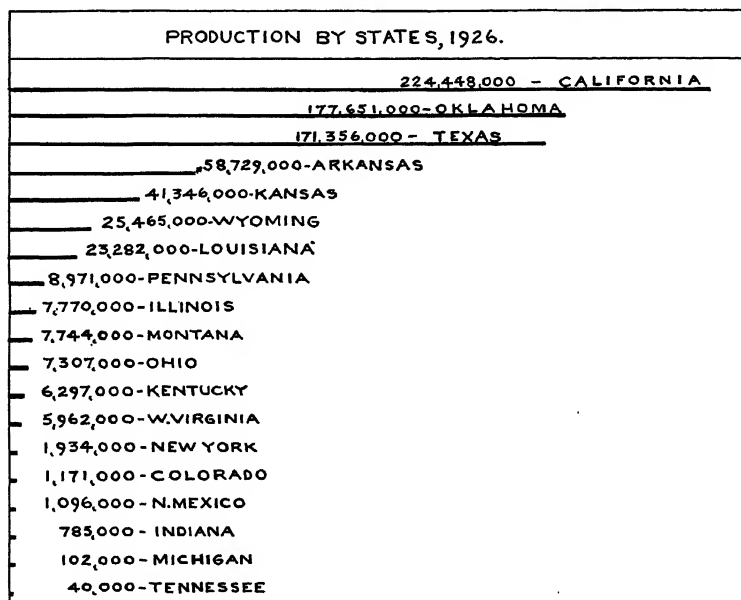


FIG. 1.—DOMESTIC CRUDE OIL PRODUCTION IN BARRELS BY STATES IN 1926.

REVIEW BY DISTRICTS

The review of the United States for 1926 is covered by the following authors for the various districts whose papers are published separately and to whom the writer is indebted for information in this general discus-

sion: California, Joseph Jensen; Oklahoma and Kansas, Henry A. Ley; Gulf Coast, Sidney A. Judson; Texas outside Gulf Coast, R. B. Whitehead; Arkansas and Louisiana, H. D. Easton, L. P. Teas, E. A. Stiller and A. F. Crider; Rocky Mountain district, Fred E. Wood; Mississippi Valley, Gail F. Moulton, W. N. Logan and R. Lee Collins; Appalachian fields, George Turnbull.

GAIN IN MAJOR POOLS, 1926 OVER 1925		LOSS IN MAJOR POOLS, 1926 UNDER 1925	
PANHANDLE	25,000,000	SMACKOVER	22,140,000
SPINDLETOP	14,309,000	WORTHAM	14,004,000
SEMINOLE	10,837,000	TONKAWA	8,856,000
WEWOKA	4,960,000	POWELL	6,583,000
URANIA	4,119,000	DOMINGUEZ	5,974,000
HUNTINGTON BEACH, GARBER, REAGAN, VENTURA	10,380,000	CROMWELL	5,442,000
TOTAL -	69,612,000	SALT CREEK	3,539,000
		BURBANK	3,450,000
		TORRANCE	2,967,000
		LONG BEACH, LULING, ROSECRANS, SANTA FE SPRINGS, ORANGE, INGLEWOOD, HAYNESVILLE	10,294,000
		TOTAL -	87,420,000

FIG. 2.—GAIN AND LOSS IN PRODUCTION OF MAJOR POOLS OF THE UNITED STATES IN 1926 COMPARED WITH THE PRECEDING YEAR.

Fig. 1 shows domestic crude oil production by states in 1926, and Fig. 2 the gain and loss in production of major pools of the United States in 1926 compared with the preceding year.

CALIFORNIA

California produced 224,000,000 bbl. in 1926 as against 230,000,000 the preceding year, a decrease of approximately 6,000,000 bbl. It easily held first place among the states but lost in relativity. In 1925 California exceeded the Oklahoma-Kansas fields by 20,000,000 bbl. as compared with only 5,000,000 last year; it produced 81,000,000 bbl. more than

Texas in 1925 but only 46,000,000 more in 1926. However, it closed the year with a production of 20,379,000 bbl. in December, which was the largest month for the twelve, and with 664,000 bbl. per day on Jan. 1, 1927, compared with 629,000 bbl. on Jan. 1, 1926. The new increase was largely due to the town lot development in the extension at Huntington Beach which passed its peak in December. The extension of the Ventura Avenue field increased its production in 1926 from 30,000 to 54,000 bbl. per day. Seal Beach, a new field, was discovered in August and with a production of 562,000 bbl. for the remainder of the year was just fairly getting under way at the beginning of 1927. Five wells were producing 10,000 bbl. per day and over 30 wells commencing. It is the opinion of Joseph Jensen that if the production situation is disturbed in California this year by a surplus of oil it will come from Seal Beach. It offers every chance of being another link in the chain of prolific pools following the Newport-Beverly trend through the Los Angeles Basin. The field contains three known sands: the Bixby at 4300 ft., the Selover at 4700 ft. and a third sand has been cored at 5200 ft. which promises production in the center of the field. Jensen estimates that the first six months of 1927 may reach 50,000 to 75,000 bbl. per day with initial production of 2000 to 3000 bbl. per well. The oil is of an average gravity of 24 with a gasoline content of 25 per cent.

The intensive development at Seal Beach will likely prevent any decline in California production for the next six months; there will be a possible increase but probably not large at Ventura Avenue. The falling off at Huntington Beach and other districts will tend to prevent a demoralizing increase at Seal Beach. The policy of the major companies who hold large proven but undeveloped reserves will depend to some extent on results outside the state but the general belief is that the production of California for 1927 will remain fairly stable at present levels. Most of the remaining undiscovered fields of the state lie in basins where the rocks are concealed by alluvium and geological exploration at the surface is of little avail; thus the defining of structural detail proceeds more slowly than heretofore; a matter which has an important bearing on the future stability of the oil industry in the United States.

OKLAHOMA AND KANSAS

Oklahoma produced approximately 177,651,000 bbl. in 1926 according to the *Oil and Gas Journal*, an increase of 1,240,000 bbl. over the final 1925 figures of the Bureau of Mines, and Kansas yielded 41,346,000 bbl., an increase of 3,062,000 bbl. Oklahoma's production exceeded that of any previous year and Kansas showed its second annual increase. Oklahoma closed the year with an average daily production of 571,045 bbl. and Kansas 118,420 bbl. as compared with 447,645 bbl. and 101,410 bbl. respectively at the beginning of the year.

New production in Oklahoma was obtained chiefly from the Wilcox sand and other sands below the "Mississippi lime" or pre-Chattanooga in age. In our discussions of last year we anticipated this fact and called attention to the enlarged possibilities of Wilcox sand production in the future. Seminole was the result of suggested inference from the discovery of production in the Wilcox sand at Wewoka, which enlarged the probable province of this prolific horizon. A wide spread continuous sand deposit comprising the Wilcox and Hominy horizons apparently underlies the entire central portion of Oklahoma stretching from the Arbuckle Mountains in the southern portion of the state northward to Kansas. Wewoka and the Seminole district lie in the southern portion with Garber and Tonkawa in the northern portion. Seventy-five miles intervene of prospective territory in which the general nature of the sub-surface and local conditions of pre-Pennsylvanian folding are yet to be determined. A few years ago, the Seminole district was thought to lie in a general basin north of the Arbuckle uplift, test wells being stopped at depths of 3700 ft. whereas drilling to 4200 ft. would have reached the Wilcox sand. The area of commercial drilling depths and of pronounced structural folding is being extended from the south portion northward and from the north portion southward.

Revision is also being made in former sand correlations which affect the general outlook for Oklahoma; for instance, late development in the Cushing pool indicates that it was not primarily a Bartlesville sand pool but furnished the main portion of its original development from the Wilcox and Siliceous Lime below the Wilcox. Recent drilling of new wells in the very heart of the old production at Cushing show that the structure was elevated at the close of Mississippian time and all beds eroded down to the pre-Chattanooga formations. Both the Wilcox and Bartlesville sands are missing at the top of the structure in places with the original development in the Siliceous Lime; in the south portion of the pool, the Wilcox produced and it now appears that the Bartlesville sand was never deposited over a considerable portion of the structure but merely flanks it on the sides chiefly in the areas of salt water. It was essentially a Wilcox and pre-Wilcox pool, whereas heretofore it had been considered a great Bartlesville field. If these facts had been known earlier a discount could have been made for pools like Tonkawa, Garber and Seminole but for years the Wilcox structures had been considered of only small local extent. Undoubtedly the future will bring forth other large pools in sands of pre-Chattanooga age, in the Misener, Hunton, Wilcox, Hominy and the Siliceous Lime throughout a broader district. A debt of gratitude is due Luther H. White for his paper delivered to the institute at the previous annual meeting and his general map which delineates the area of the pre-Mississippian formations. His work has resulted in what is perhaps the outstanding contribution to the subsurface geology of Oklahoma.

The important discovery of the year in the state was the opening up of the Seminole district including Seminole proper, Searight, Earlsboro and the "Eight-six" pools. In less than six months of active drilling in the last half of the year, Seminole rose to a production of 152,000 bbl. at the end of the year and went into the new year with a climbing production. The four pools lie near together and their extent and possible connections are yet to be determined by further drilling. Present estimates by operators in that district range from 200,000 bbl. to 350,000 bbl. per day as a peak for the district but in reality it is a wild guess at best due to the undefined limits of the sub-surface structure.

Kansas continues to furnish pools from the Pennsylvanian series which, according to Henry A. Ley, were largely responsible for the increase in that state, the new development being chiefly on and east of the Granite Ridge. The important new discoveries were the Churchill Pool in eastern Sumner County, the Kramer Pool in Butler County and the Webb Pool in Elk County.

TEXAS, OUTSIDE GULF COAST

Texas, other than the Gulf Coast, continued to be a region of great activity in drilling, leasing and prospective testing throughout the year. It is considered to offer great opportunities in development for several years to come. By some it is considered the most important district of domestic reserves. (Fohs, 1925.)

Attention was centered in 1926 on the Panhandle district, which averaged over 100,000 bbl. per day from June until the close of the year with as many as 575 wells drilling at one time. Three new areas were developed to the eastward from the original pool, a striking one of which was the Pampa district in Gray County. Two zones of production were developed, one being a lime series and the other a granite wash; the two zones parallel each other and flank the gas producing area.

The Panhandle district, west and southwest Texas, will likely receive the greater part of the attention of Texas operators during 1927. High prophecies have been made relative to the potentialities of that country. R. B. Whitehead has calculated that there is a reserve of at least three billion barrels of oil in the Panhandle and Big Lime producing areas of west Texas. He gives the north side of the Panhandle trend a reserve of 800,000,000 bbl. and expects successful results with the coming year in the development of the south flank of the present producing Panhandle trend and in other trends parallel thereto. In Crane, Upton, Reagan, Winkler and Jones counties several new pools were developed during the year which for the most part are producing from the Big Lime series similar to the producing horizon at Big Lake. None of these pools were defined at the close of the year and are considered forerunners of sizeable

pools in 1927. They are being developed on major structural trends in a northwest-southeast direction somewhat parallel to the Panhandle.

Between the Cisco-Strawn production in Brown and Coleman counties, the discovery of a field in the Cisco series in Jones County at a depth of 2500 ft. somewhat fills the gap between them and the southwest area of lime production. This appears to open up a very large territory for drilling. The discovery of new shallow production in Pecos County moved the borderland of production farther southwest.

In Central Texas including the area in and around Stephens and Eastland counties, the Ranger sand was developed in two localities, which were probably indicators of pools comparable with other districts in which oil is being produced in the Ranger sand.

The Balcones fault-line system including the Luling and Mexia-Powell districts was comparatively dormant in new production in 1926. Seventy wells were drilled north of Luling in search for additional pools. In the Mexia-Powell zone, the Nigger Creek field was the only pool discovered in the Woodbine sand in 1926. It was developed on an inner or parallel fault line west of the Mexia-Powell fault zone and operators have renewed their interest in trying to discover additional pools west of the present line of main producing fields. The Nigger Creek Pool, which made 2,083,716 bbl. in 1926, compares in importance with the Richland pool of that group. In 1926, four test wells were started to the Trinity sand below the Woodbine where it is expected at a depth of approximately 5000 ft. These tests are well positioned with reference to the fault line and will serve as regional tests from Kosse to Powell.

Mr. Whitehead has made a personal field survey in the gathering of statistics and information for his review of this district, a matter deserving the special attention of the reader.

GULF COAST

The outstanding oil development of the Gulf Coast in 1926 was the resurrection of the old Spindletop pool. Lying almost dormant for a quarter of a century, this salt dome broke forth from deeper sands in 1926, and, like the eruption of a volcano, produced 13,121,468 bbl. of oil; this being 27 per cent. of the total Gulf Coast production for the year. Nearly all of this oil came from 117 wells located on about 34 acres, or a yield of 375,000 bbl. per acre. This is roughly equivalent to 50 ft. of free oil over each acre developed. One company produced 2,300,000 bbl. from two acres, or roughly 160 ft. of oil per acre.

The total yield for the Gulf Coast in 1926 was the highest in its history; 48,885,000 bbl. or an increase of 17,930,000 over the previous year. Sidney A. Judson lists 27 pools as having produced all the oil with the exception of 10,000 bbl. Considering the relatively small acreage

involved and the limited number of pools developed, the yield is remarkable. The development of rich new pools in this area of the United States will go along for an indefinite period.

Judson states that there are now 35 known salt domes in the Gulf Coast of Texas and 17 in the coastal area of Louisiana, a total of 52. In Texas, there are 3 oil fields and 3 gas fields reported of the structural type and in Louisiana, 4 oil fields and 1 gas field of the same type. Eight of the known salt domes in Texas and 2 in Louisiana have been located by geophysical methods; 5 of these were reported in 1925 and 3 in 1926. This indicates the successful use of the seismograph and torsion balance in contributing their annual quota of newly discovered salt domes. The exploratory development of these domes is usually long drawn out in point of time; this means the number of domes under lease and development is steadily accumulating as a result of the more successful methods of their discovery. It matters not whether the salt domes on the coast (discovered and undiscovered) follow a general structural system of alignment or occur in a fortuitous manner, it appears evident that a large number are yet to be discovered. It is entirely possible that prolific, new pools on some of these salt domes might run concurrently in point of development due to the larger number under exploration. But due to the extensive drilling usually necessary to locate the oil body which frequently lies only on one flank of the dome, it is more probable that they will come under active drilling in more or less of an alternating fashion thus maintaining the production rather than greatly increasing it; this appears reasonable, at least, for the current year.

ARKANSAS AND LOUISIANA

In 1926, both Arkansas and Louisiana fell off in production; the total for the two states outside the Gulf Coast, according to H. D. Easton, was 78,711,000 bbl. or approximately the figure for Arkansas alone in 1925. Arkansas dropped 20,000,000 bbl. while North Louisiana lost 1,700,000 bbl., the two areas making respectively 58,809,000 and 19,902,000 bbl. in 1926.

The realization of commercial oil production from the Wilcox formation (Eocene) at Urania brought concentrated attention on a belt stretching through north-central Louisiana into Mississippi in which lies the Angelina-Caldwell flexure. About 150 test wells, according to L. P. Teas, were drilled in this belt in 1926. The Urania Pool in LaSalle Parish saw its maximum development. The first commercial well making oil of 22° Bé. at 1572 ft. was drilled at Urania in August, 1925, but it was not until 1926 that development got well under way. From 350 bbl. daily average in January, 1926, the production rose to 19,400 bbl.

in June and closed out the year at approximately 12,200 bbl. It had a total for the year of over 4,000,000 bbl. and covers 1940 producing acres.

Urania is as yet the only Eocene production in Louisiana outside the coastal salt dome area. Its discovery naturally led to exploratory work in search for similar pools but the results were not encouraging although the discovery of gas by the Gulf Refining Co. in Richland Parish was a result of the campaign; the gas was found from a horizon below the Urania sand, the latter lying at the top of the Wilcox formation (Eocene). The gas pool (Rayville) is from a sand probably of Tertiary age above the gas horizon in the Monroe gas field near which it is situated.

Arkansas contributed the Bradley pool in Lafayette County, Arkansas, estimated with a total of 25,480 bbl. and the Lisbon pool in Union County with 2,458,000 bbl. South of the Bradley field, near Carterville in Bossier Parish, Louisiana, a big gas well was completed in the Woodbine sand which for the first time carried production in that sand outside the Sabine Uplift proper. The district as a whole is an important producer of natural gas, as indicated by the contribution of E. A. Stiller. Renewed activity is under way in testing out deeper sands and is covered by A. F. Crider in the Arkansas-Louisiana detailed review.

ROCKY MOUNTAIN DISTRICT

The Rocky Mountain district produced approximately 37,000,000 bbl. in 1926 as against 30,000,000 bbl. in 1925, an increase of 2,000,000 bbl. due chiefly to the gain of 3,450,000 bbl. in the Kevin-Sunburst field, Montana. New fields and new sands were added to the district. In Salt Creek, Wyoming, two new wells encountered oil in the third Sundance sand which opened up reserves not previously known. The first well had an initial production of 5,386 bbl. Near Lander, in what is known as the Hudson field, a 300-bbl. well of black oil was discovered in the Tensleep, a new sand for that structure. Lost Soldier developed a well in the Sundance capable of 350 bbl. per day. Dutton Creek, a new structure, in Wyoming, furnished a small well in the Muddy sand which has not yet been tested. In the Hiawatha structure a fifty-million foot gasser was encountered in an isolated area which means a considerable gas reserve. In Utah and New Mexico there were no important new developments during the past year but a great deal of interest is being shown in the large number of drilling wells in the eastern and southeastern portion of Utah and the southeastern corner of New Mexico. The future should develop something from these territories. In the opinion of Fred E. Wood, development in the Rocky Mountain region will have to be based on seeking new types of structures or working more in detail on older areas which are more obscure at the surface. Undoubtedly new pools will be located in this territory but some difficulty will be encoun-

tered in attempting to uncover them. In Wood's opinion the production in the Rocky Mountain region will decline steadily unless new pools of major proportions are discovered.

Max W. Ball recently called attention to the fact that 100,000 bbl. daily production is shut in at Salt Creek in the Lakota sand. The wells already in the sand and tested, together with wells shut down on top of the sand on inside locations would insure this additional production. This oil is not shut in because of a lack of pipe line facilities but is held as a conservation measure in the hands of one company. Seventy wells are shut in or on top of the Lakota. The wells tested have shown initial production of 2000 to 4000 bbl. per day. The Sundance sand lies beneath the Lakota and will likely furnish large wells over a more limited area.

MISSISSIPPI VALLEY FIELDS

The fields of Illinois, Indiana, Western Ohio, Western Kentucky and Tennessee are grouped under the term Mississippi Valley fields. This district produced approximately 12,000,000 bbl. in 1926, a decline of less than 1 per cent. from the previous year. The relatively small decline of the old fields was offset by scattered new development in parts of all the states involved. According to R. Lee Collins, Tennessee increased from 22,000 bbl. in 1925 to 40,000 bbl. in 1926 and a more substantial gain is expected for the current year. The increase was due primarily to developments in the Bone Camp field. There was a considerable pick-up of drilling in Western Kentucky in Ohio, Daviess and Adair counties, the most prominent of which were the Crocus Creek area in Adair County and the Owensboro district, the latter including an eastward extension of the Aetnaville pool, development of the Ellis Pool and new shallow sand production north and northwest of Ralph.

Western Kentucky produced 1,450,000 bbl. in 1926. Indiana remained steady in production with 831,000 bbl. There was renewed activity in Pike, Daviess and Vigo counties. One of the principal features of the year was the discovery of oil in the Devonian lime at 2200 ft. in western Vigo County where there will be a stimulation of drilling in 1927.

Illinois produced 7,750,000 bbl., a decline of less than 3 per cent. due to the effect of flush production in Wabash and Clark Counties. The only new producing fields discovered in 1926 were near Friendsville in Wabash County and northeast of Jacksonville in Morgan County in areas recommended by the Illinois Geological Survey. A deep well being drilled by the Ohio Oil Co. near Bridgeport is intended to go to a depth of 5000 ft. to test the Trenton and will, if completed, be the deepest test in the state. Operators are preparing for the installation of com-

pressed air in some of the older districts which may prove a factor of considerable importance in maintaining the production of Illinois in the future.

The Lima, Ohio, field produced 1,900,000 bbl., only a very slight decline. It maintains a remarkably steady production; more so than the quartz sand fields in other portions of the Mississippi Valley.

THE APPALACHIAN FIELDS

The Appalachian fields include New York, Pennsylvania, West Virginia, Eastern Ohio, Michigan, Eastern Kentucky and Eastern Tennessee. The review for Tennessee and western Kentucky is included under Mississippi Valley fields.

The most striking feature of the Appalachian district was the development of a profitable oil pool in Michigan. In the latter part of 1925, two oil wells were completed at Saginaw, Michigan, which made a total of 1500 bbl. by the end of the year. Ninety-three wells were drilled in 1926 with a total production of 102,000 bbl. Thirty-five wells were drilling at the close of the year. The oil is found in the Berea sand at an average depth of 1840 ft. It is a high grade crude of 42° Bé. The discovery has stimulated wildcat drilling over an area in the immediate region and the future of the state is a matter to be determined. The stratified rocks are practically entirely concealed by a thick mantle of glacial drift which forbids geological exploration of the surface but the subsurface at Saginaw has proved the presence of a pronounced anticline with the production confined to the very crest. The Berea sand is over 100 ft. thick with salt water in the main body and pay sand confined to the upper 20 ft. Deeper wells are contemplated to test the possibilities of production below the present Berea wells. The Transverse and Dundee formations occur at 2300 and 2500 ft. respectively; they have showed oil in some wells on the sides of the structure and furnished the old producing field up the main basin to the southeastward in Ontario.

As in previous years, the Bradford field and the Alleghany field continued to retain all the interest in New York and northwestern Pennsylvania. There the water-flood and compressed-air methods are proving effective in a greater recovery of production. More wells were completed in these two fields than for many years past with the result that the two states showed an increase in production. New York produced 1,900,000 bbl., an increase of 260,000 bbl., while Pennsylvania produced 8,869,000 bbl., an increase of 1,038,000 bbl. No new pools were discovered in southwestern Pennsylvania, West Virginia or southeastern Ohio. In the Henry district of Clay County, West Virginia, and in Morris Township, Greene County, Pennsylvania, there was some interest centered around the discoveries of the previous year. In Central Ohio, the

Guernsey County gas field showed a highly productive development. Eastern Kentucky contributed a number of good wells in advance of production. The state produced 6,244,000 bbl., a decrease of 526,000 bbl. due to the settling of old production and a paucity of new discoveries.

THE PROSPECT FOR 1927

The potential undiscovered oil resources of the United States are so large and the amount of money being expended in search of new fields is so great that a prediction of what the production will be for a year in advance is not warranted. This will continue to be true until the production for a period of two successive years of normal business has shown a definite downward trend in the annual yield. The curve is still upward and it must hit a definitely established culminating point before estimates of approximate accuracy can be made for a year in advance. The best we can do now is to take a sort of dead reckoning like the mariner at sea in order to ascertain our latitude and longitude and determine, if possible, whither we are drifting. Production is latitude and consumption is longitude. We still have plenty of room to sail on safely. As we follow consumption in the increase of our longitude some claim to feel the cool air of approaching icebergs, but none have yet been sighted. Even when the day arrives when they are definitely on the horizon, there is abundance of room for the industry to steer around them in the wide latitude of our reserves. The ice pack itself, which bars present methods of navigation, lies north of the open sea of substitutes. The oil business has passed through frequent squalls in which its craft has heretofore, from time to time, been delayed, rocked and damaged. But the ship is unsinkable. The year 1926 was even sailing. It appears to have been a year that moved the ship farther out from the storms of the near-shore and placed the industry in safer waters. Thus it seems to have marked a trend.

Consumption has grown to the point where sustained high prices for crude are necessary to keep up production. This is not due to exhaustion of our fields but is a result of the fact that most of our shallow pools are now largely on settled production and development has moved into deeper producing territory with consequent delay in both discovery and subsequent drilling. Moreover, test wells are not started so numerously in deep, expensive areas where large undiscovered fields are to supply the future demand. Many of the drilling wells are still in shallow districts where discoveries are limited and not a material part of the main picture. Therefore the chances of discovery of several large flush fields that run concurrently in rapid development is not so great as in former years. The coincidence of the famous major pools of 1923, remarkable as it was, would be far more remarkable if repeated again. The deeper the wells

the greater the chances of delay in completing any particular well. Granting that a given number of wells start simultaneously in a pool at 4000 ft. in depth, their completion will string along one after another at wide intervals of time while the pressure of the field as a whole is being rapidly lowered by those already completed; whereas in a pool at 2000 ft., they will be completed more nearly together besides being drilled far more quickly. If Seminole at 4100 ft. had been as shallow as Cushing at 2600 ft., the production at Seminole would have been over 300,000 bbl. at a time when it was actually 100,000 bbl. and the field would have been Oklahoma's largest pool very early in its history; this granting that saturation and the gas-oil ratio are the main factors rather than pressure due to depth. Flowing of wells by compressed gas and air is now an important factor in the maintaining of flush production.

It is believed that our storage of 500,000,000 bbl. is merely the ballast in the ship that keeps it steady. It is something that can be added to or taken from with considerable elasticity without creating a panic. A reduction of stocks from a small storage would, as shown by previous experience, be a disturbing factor in the industry and tend quickly to upset its balance; on the other hand, with a steadily increasing consumption, it is reasonable to assume that it can, if necessary, stand an addition. A total of 50,000,000 bbl. of increase or decrease represents only 10 per cent. of the stocks carried. The industry can take from or add to storage at the present time without serious disturbance of profitable prices of crude to the producer and reasonable prices of refined products to the consumer. This is as it should be for a healthy industry. Herbert Hoover spoke wisely to the people when he called their attention to the fact that oil products are wonderfully distributed to the consumer as compared with the cost of other commodities. Certainly they have fluctuated within a reasonably narrow limit for a considerable period of time and are actually cheap.

The winter period of 1926-27 has shown unusual demand for gasoline, which runs somewhat parallel to the increased use of the closed type of motor car, but it is believed also that it presages a considerable growth of demand in the spring and summer months of 1927. The prospects are fair for an increase in the prosperity and general business of the world both at home and abroad. The general disturbance created by the war is rapidly becoming a thing of the past as seems to have been demonstrated from several angles in 1926. Therefore it is the belief of the writer that a prosperous year in the petroleum industry in 1927 can be looked forward to with confidence.

Exploratory development is concentrated in California, Oklahoma and Texas, which states, for the immediate future, offer the best prospects for new pools of the major class. Of 16 pools in the United States making over 10,000,000 bbl. each in 1926, 6 were in California, 4 in Okla-

homa, 4 in Texas and 1 each in Wyoming and Arkansas. Of these pools only one was a discovery of 1924 (Reagan), two were discoveries of 1925 (Panhandle and Inglewood) and two were discoveries of 1926 (Spindletop deep and Seminole). Eleven of the 16 were producing in 1923. It seems that from one to two new pools of the ten-million-barrel class are discovered on the average each year. California, Oklahoma and Texas in the order of leadership produced 75 per cent. of the total domestic production thus bearing a similar current relation to the United States as our country does to the world. Hence we must look to these states to tell the main story of 1927. In looking to the future, it should be borne in mind that in all three of the states mentioned a great percentage of the structural provinces where surface geology is most effective has already been intensively developed. The same is true to a large degree in the Rocky Mountain District. However, it is equally a fact that subsurface conditions are continually enlarging the probabilities of an immense amount of oil in areas partially or entirely concealed from structural evidence on the surface and shallow subsurface; particularly so in California and the Mid-Continent. But the situation should soon mean a slowing up of results until higher prices are capable of carrying the increased load in developing the deeper and more hidden reserves.

The event of too much overproduction this year in the face of a strong demand is still a possibility but on the other hand the few pools now rising to their peaks will be offset notably by those flush pools now on the decline, so that within the year, in order to meet the ever-increasing demand and at the same time maintain the necessary storage reserves, we may anticipate a renewed stimulus in the finding of new production.

The World's Petroleum Production during 1926

BY VALENTIN R. GARFAS,* NEW YORK, N. Y.

(New York Meeting, February, 1927)

THE world's production of petroleum during 1926 is estimated at 1,096,000,000 bbl., an increase of about 29,000,000 from that of 1925 and somewhat larger than the average during the last 4 years, which is about 1,049,000,000 bbl. The daily average during the year may be estimated at 3,000,000 bbl. of which the United States produced 2,123,000 and other countries 877,000.

The outstanding developments during the year have been the decline of the Mexican fields, the counterbalancing of this decline by the increase in Venezuelan and Russian production, and increases in Roumania and Colombia.

During 1926 the United States, Mexico and Russia retained their relative positions as the three leading producers, while Venezuela probably outranked Persia for fourth place, Roumania passed Dutch East Indies for sixth, and Colombia moved from sixteenth to the eleventh position. It further appears that should present conditions develop normally throughout 1927, it is not improbable that Mexico will be outranked by Russia as the second oil-producing country and that the increased yield of Colombia will place this country in the eighth or ninth position, and ahead of India.

The United States and Mexico should produce in 1926 about 79 per cent. of the world's total and the aggregate of the seven leading countries should be close to 95 per cent. of the total. It is interesting to note that about 85 per cent. of the total comes from fields on the American continent, the United States accounting for about 71 per cent., and Spanish-American countries and Trinidad for 14 per cent.

Production of countries other than the United States and Mexico has gradually increased during the last four years and now represents about 21 per cent. of the total.

The United States easily retains its long standing leadership with a total yield of 775,000,000 bbl., or an average daily yield of 2,123,000, representing about 71 per cent. of the total, which is about equal to that of the previous year. An increase in production of about 11,000,000 bbl. is indicated by the statistics so far available. It will be noted that the American production in the last 4 years has averaged about 746,000,000 bbl. yearly and that this uniformity, as well as the marked increases

* Manager, foreign oil department, Henry L. Doherty & Co.

of the previous years, has been sharply reflected in the corresponding world's total yearly output.

The crude and refined oils available in the United States in each year from 1920 to 1925 exceeded the demand. This resulted in the accumulation in storage of an additional surplus during the last six years, estimated at 340,000,000 bbl. The year 1926, however, shows a radically changed situation as there has been an excess of demand over supply, causing a reduction of the stocks estimated at 25,000,000 bbl. against an increase in stocks of about 27,000,000 in the preceding year. This change has been interpreted by some to represent the long expected turning of the corner, heralding a more stable situation and a brighter future for the American oil industry.

Mexican production, which during 1926 will average daily close to 248,000 bbl., declined about 25,000,000 bbl., or about 21 per cent. from the preceding year, against a decline for 1925 of about 17 per cent.

Judging by the information available regarding conditions in the Russian fields, it is estimated that the production during 1926 will be about 62,000,000 bbl., or about 170,000 daily average, representing about 5.7 per cent. of the world's total production, and an increase of about 10,000,000 bbl. over 1925.

The production of the Venezuelan fields increased from 19,700,000 bbl. to 36,500,000, the country's yield averaging about 100,000 bbl. daily, which is equivalent to $3\frac{1}{3}$ per cent. of the world's total, and as a result Venezuela will probably outrank Persia for fourth position.

The production of Persia probably increased slightly during 1926, the total being estimated at 35,400,000 bbl., a daily average of close to 97,000. It is reported that the output might have been greater were it not for the policy adopted by the Anglo-Persian Oil Co. to limit the production of wells.

The Roumanian fields should reach a new high of about 23,500,000 bbl. or about 64,400 daily average, and it is remarkable that this region, which has been producing for about 70 years, is now at its producing maximum. The country produced a little over 2 per cent. of the world's output.

The fields of Dutch East Indies continue to show a slow gradual gain, the total being estimated at 21,700,000 bbl., or close to 60,000 daily.

After 30 years of producing life the Peruvian fields should reach a new peak in 1926, estimated at 11,000,000 bbl. The country is producing about 1 per cent. of the world's total or about 30,100 bbl. daily average.

The fields of India and Poland, remained practically stationary, and there was an upward tendency in the yield of Argentina and Trinidad.

The following table shows the production of the various countries during the last three years, in the order of their importance in 1926. The figures for 1924 are those of the U. S. Geological Survey, for 1925

the estimates are those of the U. S. Bureau of Mines, and those for 1926 are the author's own estimates:

WORLD'S PETROLEUM PRODUCTION

(Thousands of barrels)

Country	1924	1925	1926
United States.....	713,940	763,743	775,000
Mexico.....	139,497	115,515	90,700
Russia.....	45,312	52,448	62,000
Venezuela.....	8,754	19,687	36,500
Persia.....	32,373	35,038	35,400
Roumania.....	13,303	16,646	23,500
Dutch East Indies.....	20,473	21,422	21,700
Peru.....	7,812	9,164	11,000
India.....	8,150	8,000	8,000
Argentina.....	4,669	5,818	6,500
Colombia.....	445	581	6,000
Poland.....	5,657	5,960	5,600
Trinidad.....	4,057	4,654	5,500
Sarawak.....	4,163	4,257	4,500
Japan.....	1,959	2,000	1,820
Egypt.....	1,122	1,226	1,250
Germany.....	406	411	600
France.....	426	459	550
Canada.....	164	318	350
Czechoslovakia.....	75	50	50
Italy.....	45	45	45
Algeria.....	12	12	12
Cuba.....	4	4	4
Others.....	109	108	109
Totals.....	1,012,927	1,067,566	1,096,690

SUMMARY

The estimates for 1926 again emphasize the controlling importance in the oil industry of the fields in the American continent, particularly those in the United States and Mexico, the figures showing further the sharp decline in Mexico and the increase in the yield of Venezuela, Colombia, Peru, Argentina and Trinidad. The situation in the European and Asiatic fields, outside of the increase in Roumanian production, which is significantly important, is somewhat indefinitely reflected, first, because of the unsettled political situation in Russia, which may interfere with the expected increase in production under normal conditions, and second, due to the absence of definite information regarding the actual reasons for the apparent lack of a larger increase in the Persian output.

The decline in American oil stocks, for the first time in 7 years, is perhaps the most significant single happening during the year.

A bird's-eye view of the world's oil industry at this time further shows the necessity for the United States, as the largest oil consumer, to conserve its present supply and to insure its future supply, within and without its territory. In this connection the recent report of the Federal Oil Conservation Board, appointed by President Coolidge, is worthy of serious consideration.

The Place of Petroleum in Industry

BY ARTHUR KNAPP,* PHILADELPHIA, PA.

(New York Meeting, February, 1927)

THE petroleum industry is confronted with the problem of conservation. This is not a simple problem in view of a wide difference of opinion among authorities as to whether conservation is necessary and the specific direction that it should take. Many writers accuse the petroleum industry of wasting the natural resources of the country. Others link the consuming public with the industry in this waste.

Let us define our terms; agree, if possible, on where and when waste occurs in connection with either the recovery or use of petroleum and select such plans for conservation as appear most practical. A plan for conservation which appears to be logical today may not meet the problem tomorrow. The conservation of petroleum is closely linked with the advancement of many arts and sciences and as these advance, the plan of conservation has to change.

The need for conservation does not necessarily rest upon the elimination of waste. There is not necessarily a fixed connection between waste and conservation. Conservation may be necessary or desirable although there is no waste.

DEFINITIONS

Waste: Useless or unnecessary expenditure.

Conservation: The preservation of the natural resources for economical use.

Both definitions contain adjectives the interpretation of which is extremely difficult. It is the determination of what expenditures are "unnecessary" or what uses are "economical" that will require study and agreement. If one lets his automobile engine run in cold weather in order to keep it hot, it is difficult to measure that necessity. If an automobile is driven two blocks, when the driver is physically capable of walking, what is the measure of necessity. This question is in the realm of one of the oldest of sociological problems, "What is a luxury?"—a problem which has not been solved. The history of society indicates that the measure of a man's wealth has been the measure of necessity. What he can afford to buy without stinting himself in some other direction, is not a luxury to him and, if it fulfills his desires, may be considered his necessity.

* Petroleum engineer, United Gas Improvement Co.

As a matter of fact, dollars and cents are the final measure of "necessity" and eliminate waste. Heat exchangers are not used in a refinery because of the "unnecessary" waste of heat in water condensers but because it is profitable. Central electric generator-plants are not taking the place of individual steam plants in the oil fields because of the "unnecessary" use of fuel oil, but because they are profitable. The practical solution of the problem of the elimination of waste appears to be through the operation of the simple economic law of profit through economy.

CONSERVATION

The desire or necessity for the "preservation of natural resources for economical use" depends upon the individual opinion of what constitutes "economical use." Conservation is subject to the same analysis as waste. Generally, however, conservation implies a study of future possibilities, both as to supply and demand. The practical application of conservation means the practice of certain economy of use at the present time, in order to reap profit in the future. Sentimental consideration of the needs of future generations does not appear practical. It is impossible to conceive what specific needs they will have or to measure these needs. All future generations ask is that we build well, that we do whatever we do to the best of our ability.

Conservation of natural resources is not entirely a matter of the individual. Insofar as the resources affect the welfare and security of the nation as a whole, just so far is the whole nation interested in the problem. This problem should not be projected too far into the future. It should be solved as a problem of today and the welfare and security of the nation will be cared for in the future, as it has been in the past, by the advancement of science and industry. Conservation is, in a measure, brought about by the elevation of a commodity to a higher plane of use. As an illustration, petroleum used in an internal combustion engine to generate an electric current—which current is in turn used for lighting—is in a higher plane of use than kerosene used in a lamp.

It is necessary to realize the position of petroleum in industry in order to realize how serious is the problem of conservation and of the prevention of waste. Conservation cannot be solved as an academic problem but as a practical problem that will meet the greatest needs of the largest number of users.

PETROLEUM AS A SOURCE OF ENERGY

Petroleum products are fundamentally energy sources and the position which they occupy among energy sources is of primary importance. (Table 1.)

Tryon and Mann¹ have made the following estimate of energy sources for 1923:

	B.t.u. (in Trillions)	Percentage of Total
Coal.....	17,000	65
Domestic oil.....	4,440	16
Natural gas.....	1,080	4
Imported oil.....	490	2
Water power.....	1,140*	4
Work animals.....	850*	3
Firewood.....	1,500*	6
Wind, sun, tides, waves.....	Negligible	

* Heat value of coal necessary to produce the same result.

TABLE 1.—*Percentage of Total B. t. u. Equivalent Contributed by the Several Mineral Fuels in the United States 1819 to 1925*

Prepared by F. G. Tryon, Bureau of Mines, Department of Commerce

Year	Anthra- cite	Bitumi- nous	Total Coal	Domestic Oil	Natural Gas	Imported Oil	Total Oil and Gas	Grand Total Fuels	*Water Power Per Cent. of Fuels
1819	100.0		100.0					100.0	No data
1829	58.4		100.0					100.0	No data
1839	40.2	59.8	100.0					100.0	No data
1849	62.8	37.2	100.0					100.0	No data
1859	62.3	37.7	100.0	^b				100.0	No data
1869	51.3	45.9	97.2	2.8	^b		2.8	100.0	No data
1879	42.5	51.3	93.8	6.2	^b		6.2	100.0	No data
1889	29.3	59.3	88.6	5.0	6.4 ^c		11.4	100.0	2.2
1899	22.5	69.5	92.0	4.7	3.3 ^c		8.0	100.0	1.9
1909	16.0	72.2	88.2	8.0	3.8	^b	11.8	100.0	3.0
1913	14.4	72.7	87.1	8.7	3.6	0.6	12.9	100.0	3.4
1918	12.8	72.3	85.1	10.1	3.7	1.1	14.9	100.0	4.0
1919	13.3	67.9	81.2	12.6	4.5	1.7	18.8	100.0	5.0
1920	11.4	69.3	80.7	12.4	4.0	2.9	19.3	100.0	5.0
1921	14.0	61.7	75.7	16.0	4.0	4.3	24.3	100.0	5.1
1922	8.5	63.3	71.8	19.1	4.7	4.4	28.2	100.0	5.9
1923	10.9	63.5	74.4	18.9	4.6	2.1	25.6	100.0	4.9
1924 ^d	11.4	60.2	71.6	20.4	5.8	2.2	28.4	100.0	5.5
1925	7.9	63.6	71.5	21.1	5.7	1.7	28.5	100.0	6.0

* No figures for water power are available prior to 1889. Hence to make the figures comparable, the data for water power thereafter are expressed as percentage of the fuel total, but are not included in the base on which the percentage is computed.

^b Less than 0.1 per cent.

^c Figures for 1869 to 1899 based on estimated quantity of coal displaced.

^d Preliminary figures.

¹ F. G. Tryon and L. Mann: Mineral Resources for Future Population. (Pollak Foundation) 123.

The abstract measure of the position of petroleum as an energy source does not place it in as important a place as might be expected. A decrease of one-quarter of the domestic oil available would mean a redistribution of but 4 per cent. of the total energy in use. The economic value of petroleum products does not rest entirely upon their latent energy value. The convenience of this form of energy is the principal factor; that is, relative concentration, ease of handling, price, width of distribution, etc. The problem of the redistribution among other sources of energy, of the energy load born by petroleum, requires the development of other sources up to a point of convenience in use which would be comparable to petroleum products.

Redistribution of energy sources is constantly going on. This constant redistribution, coupled with changes in methods of use and increased economy of use, precludes either a shortage of any one form of energy or an excessively high price for that energy. Any decrease in available petroleum will be taken care of by the development of new forms of convenient energy and new methods for the transformation of energy into useful work.

The reserves of energy, as represented by petroleum, are estimated to be 7 billion bbl. and that of coal 3527 billion tons. On an energy basis this is a ratio of approximately 1680 to 1. Any prediction of a shortage of petroleum should not be taken to mean a shortage of energy.

Because coal and oil are the most convenient forms of energy at the present time does not indicate that they are the ultimate in useful energy. It is entirely possible that the sun, wind, tides and waves will all furnish vast quantities of energy in the future and the reserves of these sources are infinite.

Cost of Petroleum Energy

The cost of petroleum products for the production of heat has compared favorably with other energy sources up to 1926. As consumption has now overtaken production of petroleum, the rise in the price of petroleum products will be more rapid in the future.

Table 2 shows the relative prices of latent energy. The relative cost of useful work depends upon the efficiency of the transformation of this energy into useful work. Rapid strides are being made with all sources of energy and with the development of new forms of energy in intermediate stages between the original coal or petroleum and the transforming engine. These are known as synthetic fuels. The future possibilities for the more efficient use of all forms of energy are very great. We can scarcely guess which path of development will lead

to the greatest economy of transformation and what will be the ultimate form of fuel and engine or the ultimate economy in use.

TABLE 2.—*Indexes of Prices of Various Forms of Energy, 1913 to 1926^a*

Date	Fuels 20 Commodities	Domestic Fuel	Domestic Light	Bituminous Coal	Coke	Crude Oil	Fuel Oil	Average Retail Lighting Rates
1913	100	100	100	100	100	100	100	100
1914	93			93	74	86	73	
1915	88			91	73	62	57	92
1916	126			150	133	135	97	92
1917	169			264	338	190	169	93
1918	170			210	245	235	209	91
1919	181			211	194	244	124	90
1920	241			459	442	364	291	92
1921	199			270	149	182	76	91
1922	218			295	290	193	105	90
1923	185	197	123	224	224	154	103	89
1924	172	182	121	169	148	155	106	87
1925	172	181	121	167	168	179	122	86
1926 ^b	179	179	118	154	116	219	203	85

^a From *Survey of Current Business* of Department of Commerce.

^b June.

COST OF PETROLEUM AS A COMMODITY

W. S. Farish, president of the Humble Oil & Refining Co., gives some interesting figures on the cost of gasoline based upon the purchase value of the dollar:

YEAR	EXCHANGE VALUE, GALLONS-CENTS	YEAR	EXCHANGE VALUE, GALLONS-CENTS
1913	16.8	1920	13.0
1914	14.7	1921	17.8
1915	13.7	1922	16.9
1916	18.1	1923	13.5
1917	13.4	1924	12.0
1918	12.5	1925	12.1
1919	11.9		

The following index numbers are from the *Survey of Current Business* of the Department of Commerce. (Table 3.)

TABLE 3.—*Indexes of Cost of Living and Commodities, 1913 to 1926*

Year	All Commodity Price Index	Cost of Living	Pig Iron	Mid- Continent Crude	Gasoline, New York	Fuel Oil, Oklahoma
1913	100		100	100	100	100
1914	98	100	87	86		73
1915	101	101	93	62		57
1916	127	109	132	135		97
1917	177	131	259	190	141	169
1918	194	159	215	235	144	209
1919	206	172	189	244	146	124
1920	226	198	280	354	174	291
1921	147	167	157	182	155	76
1922	149	157	168	193	149	105
1923	154	161	186	154	123	103
1924	150	164	141	155	107	106
1925	159	168	135	179	114	122
1926*	152	167	123	219	125	136

These figures indicate that petroleum products, as commodities, have been and still are relatively low in cost.

As consumption has practically overtaken the supply of petroleum products, it follows that the price will have to increase until some users are eliminated or until economy of use is enforced. It is difficult to say just how high prices will have to rise to bring about these changes. Fuel-oil users are already turning to other forms of energy or to the practice of greater economy of use. The consumption of fuel oil for the generation of steam in public utility power-plants decreased from 15,000,000 bbl. in 1923, and 17,000,000 bbl. in 1924, to 10,000,000 bbl. in 1925, and an estimated consumption of less than 11,000,000 bbl. in 1926. The railroads consumed a smaller volume of fuel oil in 1925 and 1926 than in 1924.

THE PETROLEUM INDUSTRY AS AN INDUSTRY

The capital invested in the petroleum industry is approximately \$10,000,000,000, according to W. S. Farish. This is an enormous investment when it is considered in what a short time this capital has been accumulated. The railroad investments are estimated by the Interstate Commerce Commission to be \$22,000,000,000. The electric power-plant investment is estimated to be \$6,493,000,000.

The petroleum industry is one of the greatest producers of wealth. The Bureau of Mines estimates the total value of crude oil at the wells in 1925 to have been \$1,284,960,000 and in 1924, \$1,022,683,000.

The value of the output of the natural gas industry, an integral part of the petroleum industry, is estimated as follows and is compared to the value of the output of manufactured gas:

VALUE OF OUTPUT OF NATURAL AND MANUFACTURED GAS

Value in Millions of Dollars					
Date	Natural Gas				Manufactured Gas Total
	Domestic Consumption	Industrial Consumption	Carbon Black	Total	
1924	154	100	116	370	
1923	142	98	117	357	392
1922	127	95	58	280	
1921	110	64	54	228	372
1920	109	87	40	236	
1919	88	72	38	198	282

According to the U. S. Bureau of Mines the value of the annual output of natural gas gasoline, 1917-1925, was as follows:

1925	\$120,383,000	1920	\$71,788,000
1924	82,233,000	1919	64,197,000
1923	77,268,000	1918	50,364,000
1922	72,711,000	1917	40,189,000
1921	61,815,000		

The transportation of petroleum and petroleum products furnished 4.8 per cent. of the total railroad freight tonnage in 1925, 3.84 per cent. in 1924, and 3.58 per cent. in 1921. The gross tonnage of tankers on June 30, 1925, was reported by the Department of Commerce as 2,411,719 tons. This is approximately 14 per cent. of the total tonnage of vessels engaged in foreign trade under the American flag. The tonnage, classified as petroleum products, that passed through the Panama Canal was 19.5 per cent. of the total in 1915 and 28 per cent. in 1924. Thirty-five per cent. of the world's gross tonnage of steel vessels is either oil-burning or motor-driven ships.

As a tax-revenue producer, the petroleum industry stands well to the front. Gasoline taxes for 1925 amounted to nearly \$150,000,000. The total tax revenues from state registration fees, Federal excise taxes and state gasoline taxes were as follows: 1924, \$544,000,000; 1923, \$382,000,000; 1922, \$289,000,000, and 1921, \$245,000,000. These taxes do not include the enormous sums paid for local taxes, income taxes, manufacturing taxes, etc.

The total taxes paid by Class A railroads were: 1925, \$358,294,000; 1924, \$380,522,000; 1923, \$339,577,000; 1922, \$308,145,000; and 1921, \$283,163,000.

The petroleum industry and the automotive industry are so closely related that their economics are interrelated. The capital invested in

automotive manufactures is approximately \$1,750,000,000. The wages paid in the automotive industry in 1923 were \$597,000,000, in the iron and steel industry, \$697,000,000, in the wholesale meat-packing industry, \$167,569,000 and in the refining industry, \$103,834,000. The automotive industry consumes approximately the following percentage of the total output of the industries listed: iron and steel, 14 per cent.; aluminum, 50 per cent.; plate glass, 50 per cent.; leather and upholstery, 70 per cent.; and rubber, 80 per cent.

The wholesale values of output of various industries, in 1923, according to the Census of Manufactures, were as follows:

1. Motor vehicles.....	\$3,163,328,000
2. Steel and iron.....	3,154,325,000
3. Slaughtering and meat packing.....	2,585,804,000
4. Foundry and machine shops.....	2,337,808,000
5. Cotton goods.....	1,901,126,000
6. Petroleum refineries.....	1,793,700,000

Petroleum is an important factor in foreign trade. . In 1925, 9.25 per cent. of the total exports of the country and 2.5 per cent. of the total imports were petroleum or petroleum products.

TREND OF SUPPLY AND DEMAND

The remarkable growth of the petroleum industry is illustrated by the following tables (Tables 4 and 5) of indexes:

TABLE 4.—*Indexes of Consumption of Various Products*

Year	Crude Oil	Gasoline	Iron Ore	Bituminous Coal	Cotton
1917	100	100	100	100	100
1920	137	162	98	96	81
1921	140	174	38	74	79
1922	159	202	72	81	89
1923	194	250	99	98	96
1924	205	296	74	92	81
1925	235	357	91	95	94

TABLE 5.—*Indexes of Production of Various Products*

Year	Crude Oil	Gasoline	Pig Iron	Cement	Bituminous Coal	Cotton	Beef
1913	100	100	100	100	100	100	100
1920	178	279	120	108	119	105	119
1921	189	300	54	107	87	97	113
1922	224	355	87	131	85	109	126
1923	295	430	130	151	118	117	130
1924	287	512	101	164	101	99	133
1925	304	620	118	177	109	115	139

CONCLUSION

The foregoing is not for the purpose of proving the position of petroleum in industry, but to emphasize it. The object of this paper is to bring out forcibly the broad aspect of the problem of conservation. The tendency has been for each branch of the industry to "pass the buck." Each branch must set its own house in order and it is going to take a degree of broadmindedness, heretofore unknown in the oil business, to bring about any agreement on the form of conservation needed and the methods to be followed.

The Trend of the Petroleum Situation

By JOSEPH E. POGUE,* NEW YORK, N. Y.

(New York Meeting, February, 1927)

THE outstanding features of 1926 in the petroleum industry included the following:

1. Stocks of all oils (crude and refined) were reduced, making the first annual decline since 1918.

2. Domestic crude oil production increased steadily during the last 11 months of the year, accelerating during the fourth quarter, and ending the year at the highest level ever attained.

3. Three new major oil fields were developed—Panhandle, Spindletop and Seminole.

4. Important advances were registered in production technique, tending to retard the declines of old wells and to maintain the flush periods of new wells.

5. The problem of gaining greater economic control of production through unit operations and gas conservation received growing recognition by the industry and the Federal Oil Conservation Board.

6. The rate of drilling was sharply increased during the second quarter of the year and suffered only a very slight retardation during the closing months.

7. The runs of crude oil to stills were maintained above normal throughout the year and underwent acceleration during the fourth quarter.

8. The gasoline situation was marked by a continuous and substantial increase in the rate of production of cracked gasoline, and by an improvement in the statistical position of the commodity.

9. Prices of both crude petroleum and its principal derivatives were characterized by more profitable levels and greater stability than for a number of years past; and, while showing declines during the fourth quarter, prices did not recede sufficiently to slow down materially either drilling or refinery operations.

CHANGE IN STOCKS

The keynote of 1926 was a reduction in stocks of all oils to the extent of approximately 24,000,000 bbl., or 4.4 per cent., of which roundly

* Consulting engineer.

8,000,000 bbl. represented a fire loss in California, leaving 16,000,000 bbl. as a measure of the deficiency of current supply. This reduction in stocks, while moderate in view of the 518,000,000 bbl. still on hand, is nevertheless worthy of note following a 7-year period of growing inventories, as shown in Table 1.

TABLE 1.—*Supply of, Demand for, and Stocks of All Oils in the United States by Years, 1918–1926*

Data largely from Federal Oil Conservation Board and U. S. Bureau of Mines
Stocks Calculated by Author

Year	Supply, Millions of Bbl.	Change, Per Cent.	Demand, Millions of Bbl.	Change, Per Cent.	Stocks, (Approximate) Millions of Bbl.	Change in Stocks, Milli- ons of Bbl.
1918	395	420	161	— 25
1919	433	+ 8.4	426	+ 1.3	168	+ 7
1920	552	+27.3	524	+22.9	196	+ 28
1921	601	+ 8.9	517	— 1.2	280	+ 84
1922	693	+15.0	591	+14.2	382	+102
1923	832	+20.0	731	+23.7	483	+101
1924	808	— 2.9	777	+ 6.1	514	+ 31
1925	834	+ 3.0	806	+ 3.7	542	+ 28
1926 ^a	851	+ 2.1	865 ^b	+ 7.2	520	— 22 ^c

^a December estimated.

^b This figure does not include 8,000,000 bbl. fire loss.

^c Includes 8,000,000 bbl. fire loss.

It may be noted from Table 1 that in 1926, as compared with 1925, the supply of all oils increased 2.1 per cent., while the demand for all oils increased 7.2 per cent.

The rate of change in stocks of all oils by months from 1922 to the present is shown in Fig. 1. This diagram illustrates the marked improvement in the statistical position of the industry that characterized the latter part of 1925 and nearly all of 1926, as compared with the preceding period. The closing months of 1926, however, may be observed to have undergone a progressive impairment in position as compared with the earlier part of the year.

PRODUCTION

Production of crude petroleum in the United States in 1926 declined during the early winter, increased steadily during the spring and summer, and mounted sharply during the fall, ending the year at approximately 2,400,000 bbl. per day, the highest level ever attained. (See Fig. 2.) For the year as a whole the output totalled about 767,000,000 bbl.,

an increase of 11,000,000 bbl., or 1.3 per cent., over the quantity produced in 1925. As shown in Fig. 2, the 1926 output ran below the 1925 levels during the first 7 months of the year, but forged progressively ahead thereafter.

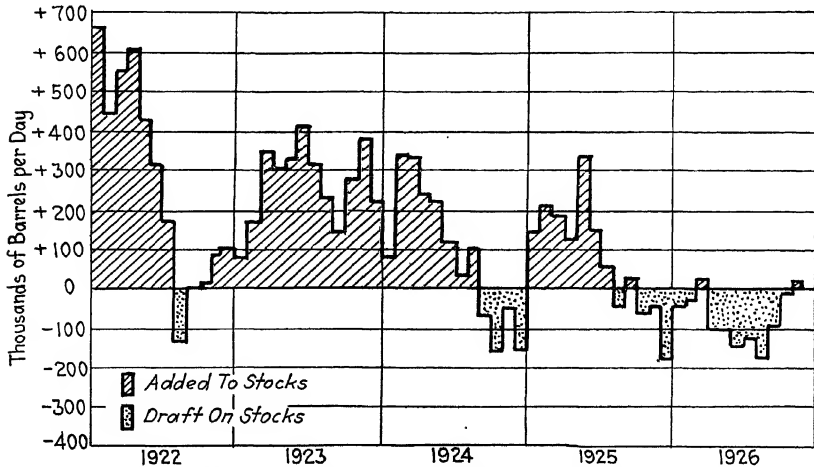


FIG. 1.—RATE OF CHANGE IN THE STOCKS OF ALL OILS IN THE UNITED STATES BY MONTHS, 1922-1926.

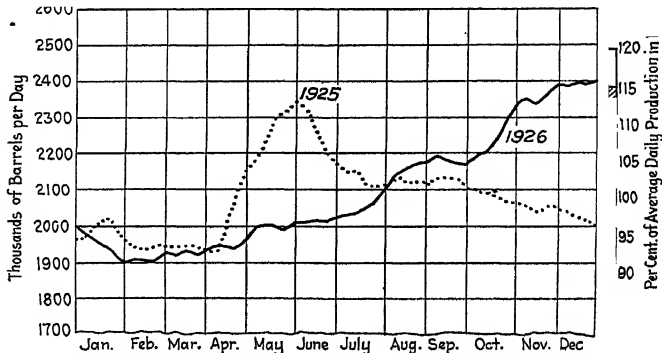


FIG. 2.—DAILY RATE OF GROSS PRODUCTION OF CRUDE OIL IN THE UNITED STATES IN 1926 COMPARED WITH 1925.

The marked expansion in production was caused primarily by the development of three major fields, the exploitation of a much greater number of smaller pools, and the application of improved production technique that operated to retard the normal decline of old wells. The part played by the three major fields—Panhandle, Spindletop and

TABLE 2.—*Well Data Covering the Oil Fields of the United States (Exclusive of California) by Years, 1913-1926*Data from *Oil and Gas Journal*

Year	Number of Oil Wells Completed	Total Initial Daily Production, Bbl.	Average Initial Daily Production per Well, Bbl.	Number of Rigs Up and Wells Drilling End of Year
1913	18,824	781,000	42	3,102
1914	16,140	1,471,000	91	1,953
1915	8,892	1,715,000	193	2,835
1916	18,080	1,497,000	83	3,956
1917	15,673	1,383,000	88	4,908
1918	16,698	1,486,000	88	6,360
1919	20,686	3,366,000	163	9,699
1920	26,738	3,388,000	127	8,748
1921	13,943	2,640,000	189	6,148
1922	16,201	3,912,000	241	5,339
1923	15,327	3,917,000	255	4,029
1924	13,533	2,747,000	203	4,632
1925	15,611	3,793,000	243	4,707
1926	18,032	3,299,000	183	5,891

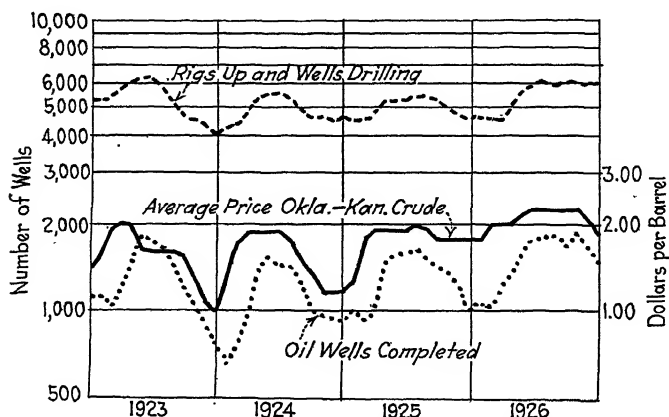


FIG. 5.—TREND OF DRILLING OPERATIONS EAST OF CALIFORNIA BY MONTHS, 1923-1926, SHOWING THE HIGH DEGREE OF CORRELATION WITH PRICE, AND THE MOMENTUM MAINTAINED AT THE CLOSE OF 1926.

drilling operations were only moderately retarded by the price declines in the closing months of 1926, and 1927 was entered with relatively considerable drilling momentum.

REFINERY OPERATIONS

The best criterion of refinery operations is the monthly volume of crude oil charged to stills. The actual figures as reported by the U. S. Bureau of Mines, however, are rendered more significant if adjusted, by appropriate methods, for the seasonal and growth elements and thus converted into an index expressing the runs to stills in percentages of computed normal. This procedure has been followed and the results are presented graphically in Fig. 6. This diagram indicates that for the past two years refinery operations have ranged substantially above computed normal, thus correlating with business activities in general and reflecting the larger demands prevailing for petroleum products.

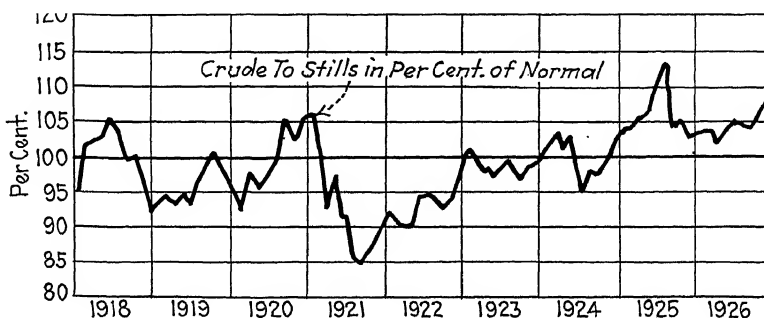


FIG. 6.—INDEX OF REFINERY OPERATIONS IN THE UNITED STATES BY MONTHS, 1918-1926.

Refinery activities have shown an upward trend during the latter three-quarters of 1926, with a sharp upturn during the closing months of the year. The year 1927 is entered with refinery operations (as well as drilling and production) displaying a relatively high degree of momentum.

GASOLINE

The outstanding features of the gasoline situation in 1926 were: the maintenance of a remarkably large demand, averaging about 42,000,000 bbl., or 17.3 per cent., over 1925; an important increase in the output of cracked gasoline, amounting to 22,000,000 bbl. or 32 per cent., over the production of this type of gasoline in 1925; and a slight reduction in the quantity of gasoline held in storage. These features are shown in the following table:

TABLE 3.—Gasoline Data in 1925 and 1926

Year	Production Cracked Gasoline, Million Bbl.	Production Other Gasoline Plus Imports, Million Bbl.	Total Supply, Million Bbl.	Domestic Consumption, Million Bbl.	Exports, Million Bbl.	Total Demand, Million Bbl.	Change from Previous Year, Per Cent.	Stocks End of Year, Million Bbl.
1925	69	193	262	224	32	256	+19.4	39.3
1926 ^a	91	206	297	254	44	298	+17.3	39.1

^a December estimated. Subject to revision.

The rate of growth in the demand for gasoline over the last 10 years has been a remarkable feature of the oil business. During this period, the annual rate of growth, determined by fitting a compound interest curve to the data by the method of least squares, has been 17.5

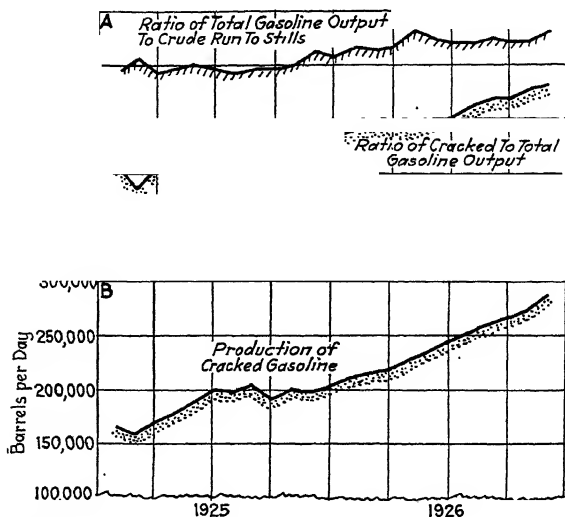


FIG. 7.—TRENDS OF SIGNIFICANT ELEMENTS IN THE GASOLINE SITUATION BY MONTHS, 1925-1926. A. PERCENTAGE RECOVERY OF GASOLINE FROM CRUDE OIL AND PROPORTION OF TOTAL GASOLINE OUTPUT CONTRIBUTED BY CRACKING. B. DAILY RATE OF PRODUCTION OF CRACKED GASOLINE BY MONTHS, 1925-1926.

per cent. The equation of this fitted curve is: $Y = 68.0 (1.175)^x$, origin at 1917, unit = 1 million barrels. The data together with the fitted curve are shown in graphic form in Fig. 11.

The trend of the output of cracked gasoline is shown by months for 1925 and 1926 in Fig. 7B, while the relation of this factor to the total supply of gasoline and the ratio of total gasoline production to the

volume of crude oil charged to stills is displayed in Fig. 7A. It is seen that the increase in output of cracked gasoline has been substantial and progressive and a growing proportion of our crude oil supply is being converted into gasoline.

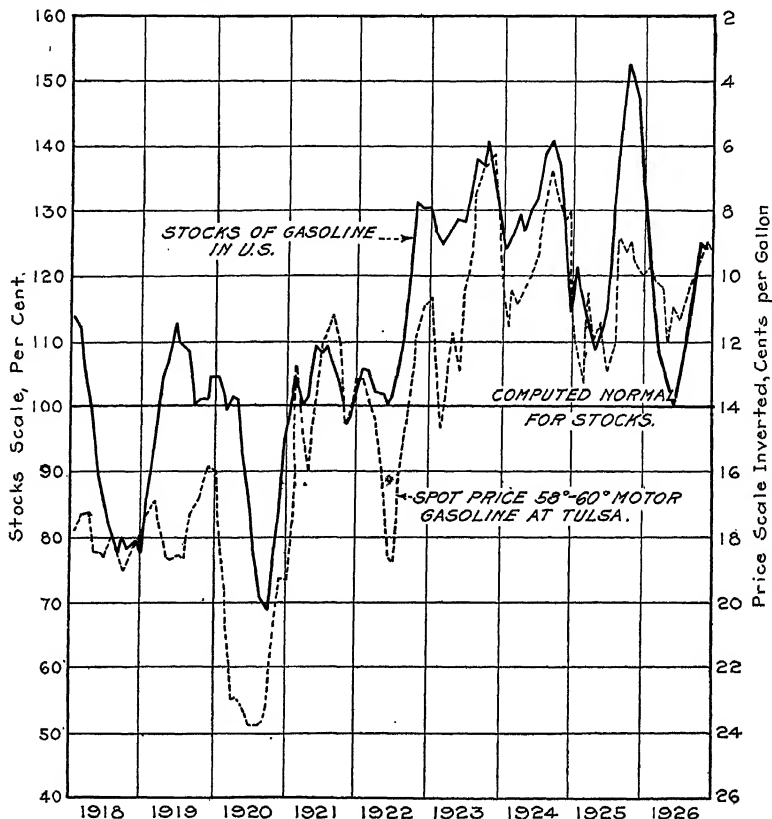


FIG. 8.—CORRELATION BETWEEN STOCKS OF GASOLINE IN THE UNITED STATES (EXPRESSED IN PERCENTAGES OF COMPUTED NORMAL) AND THE SPOT PRICE OF MOTOR GASOLINE AT WHOLESALE ON THE TULSA MARKET, BY MONTHS, 1918-1926.

A further interpretation of the gasoline situation is presented in Fig. 8 which shows by months from 1918 to the present, the correlation existing between stocks of gasoline in the United States (corrected for seasonal and growth) and the spot price of 58-60 motor gasoline on the Tulsa market. This correlation, and the technique underlying it, was presented¹ at the 1925 annual meeting of the Institute, and it is here

¹ J. E. Pogue: The Relation between Price and Stocks of Gasoline. Petroleum Development and Technology in 1925, 732.

shown with the 1926 data added. It may be observed that the trend of the gasoline situation since 1923 has been in the direction of statistical improvement, although the situation lost strength during the second half of 1926 and 1927 was entered with a gasoline carry-over above "normal," though relatively smaller than the carry-over into 1926.

During the year a new element made its appearance in the gasoline situation, which may have some bearing upon both future supply and price. This new element arises out of competition for the favor of the consumer, and is taking the direction of improvement in the quality of the product. A number of companies, responsible for a sizable proportion of the country's gasoline output, have standardized on a higher-gravity lower-endpoint product which is designed to improve the quality, if not the efficiency, of performance in the motor. The extent to which this movement will go is not yet clear, but the effect of the tendency is to cut down the percentage of straight-run gasoline derivable from a barrel of crude oil and probably also to add a further stimulus to the growth of cracking.

FUEL PRODUCTS

The outstanding economic factors in respect to kerosene and fuel oil are found in the ratios of the production of each of these products to the volume of crude oil charged to stills. As shown in Table 4, the percentage recovery of these two products has for several years shown a

TABLE 4.—*Trend of Production, and Percentage Recovery, of Kerosene and Fuel Oil by Years, 1922-1926.*

Year	Crude Oil Run to Stills, Million Bbl.	Production		Recovery per Barrel of Crude to Stills	
		Kerosene, Million Bbl.	Fuel Oil, Million Bbl.	Kerosene, Per Cent.	Fuel Oil, Per Cent.
1922	501	54.9	255	10.9	50.8
1923	581	55.9	287	9.5	50.2
1924	644	60.0	320	9.3	49.7
1925	740	59.7	365	8.1	49.4
1926*	781	61.7	364	7.9	46.6

* December estimated.

downward trend. The cause of this changing relationship is primarily the growth of cracking and secondarily progressive improvements in the efficiency of fractionation. The recovery factors for kerosene and fuel oil, therefore, bear a reciprocal relation to the recovery factor for gasoline.

The development of fuel oil through cracking as a second important source of gasoline has brought a revolutionary economic force into the petroleum situation. For the period that cracking can continue to

expand adequately, this factor has divorced the price of gasoline from the price of crude oil (see Table 6) and set up fuel oil as the great equalizing element between the supply of crude oil and the demand for gasoline. Thus the economics of petroleum rests upon a triangle of forces, which are interdependent and to a large degree automatic in adjustment. When there is a deficiency of crude oil, crude oil prices advance, profits in making cracked gasoline become larger, and cracking plants are built in growing numbers until the demand for gasoline is met from relatively less crude petroleum; there being a corresponding decrease in the output of fuel oil, whose elastic demand contracts proportionally. When the crude oil supply, on the contrary, becomes too large, crude oil gets cheaper, the profit incentive for expanding cracking operations loses force, and more crude oil is converted into fuel oil, whose demand is quickly responsive to a lowering of price. These relationships will serve to interpret many of the price movements of the past few years.

EXPORTS

An important feature of the oil business in 1926 was the substantial increase in exports, which showed a gain of 18,400,000 bbl., or 16.8 per cent., over 1925. The trend of the principal elements composing the export movement is shown in Table 5, where it appears that the chief component in the gain was gasoline, which showed an increment of 12,500,000 bbl. or 39.4 per cent. The large overseas movement of gasoline was an important element in maintaining domestic prices for this commodity.

TABLE 5.—*Exports of Oil from the United States by Years, 1922–1926 (in Millions of Barrels)*

Year	Crude Oil	Gasoline	Fuel Oil	Other Products	Total, All Oil
1922	10.8	14.2	18.5	29.4	72.9
1923	17.5	20.8	33.4	28.7	100.4
1924	18.2	29.0	37.2	31.1	115.5
1925	13.4	31.7	36.2	30.5	111.8
1926*	15.2	44.2	38.0	32.8	130.2

* December estimated. Figures subject to revision.

During the closing months of 1926, the large water movement of oil resulted in a shortage of tankers and a consequent rise in tanker rates, with important results in oil markets.

PRICES

One of the striking features of 1926 in the petroleum industry was the progress gained in the direction of price stabilization. This trend

is graphically indicated by Fig. 9, which shows for the last 5 years monthly index numbers representing the weighted average price of crude petroleum and petroleum products in the United States. It may be readily observed that the amplitude of the price cycle was less in 1925 than in 1924, and still less in 1926 than in 1925.

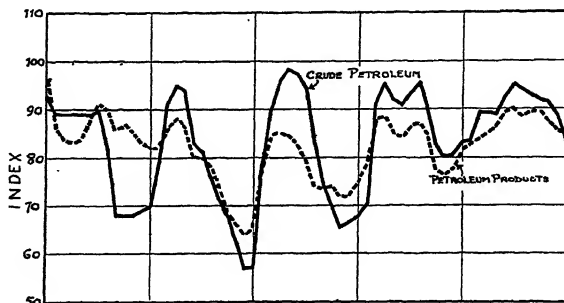


FIG. 9.—RELATIVE PRICES OF CRUDE PETROLEUM AND PETROLEUM PRODUCTS IN THE UNITED STATES BY MONTHS, 1922-1926, ILLUSTRATING THE TREND TOWARD PRICE STABILIZATION DURING THE PAST TWO YEARS. (PRICES IN 1919 = 100.)

A more detailed analysis of price movements during 1926 is afforded in perspective by Table 6 and Fig. 10. The tabular and graphic data there presented represent a continuation of the index numbers of oil prices previously published by the author.² Table 6 indicates that oil prices in 1926 averaged higher than in 1925, the percentage changes running as follows:

	PER CENT.
Crude oil	. + 3.5
Gasoline	. + 5.8
Kerosene	. +19.9
Fuel oil	. + 0.8
Lubricants	. 0.0

Fig. 10 shows that oil prices tended to advance during the first half of the year, displayed relative stability during the third quarter, and receded during the fourth quarter. The declines, however, were less severe than the corresponding recessions of a number of previous years.

In terms of the general price level of all commodities, the average prices of gasoline and fuel oil during 1926 stood at 76 per cent. and 91 per cent. respectively, while the prices of crude petroleum, kerosene and lubricants averaged 118 per cent., 124 per cent. and 113 per cent., respec-

² J. E. Pogue: The Trend of Prices in the Petroleum Industry. Petroleum Development and Technology in 1925. See pp. 738-744 for explanation of index numbers.

TABLE 6.—*Trend of Relative Prices of Crude Petroleum and Its Derivatives in the United States, by Years from 1913 to 1926 Inclusive, and by Months, 1926*

Prices in 1913 = 100

Year	Crude Petroleum at Wells	Gasoline, Tank-wagon	Kerosene, Tank-wagon	Fuel Oil at Refinery	Lubricating Oils Jobbing Quotations	All Commodities U. S. Bureau of Labor Statistics
1913	100	100	100	100	100	100
1914	82	83	97	85	101	98
1915	66	75	90	68	97	101
1916	117	121	101	98	119	127
1917	155	132	108	147	126	177
1918	195	139	130	189	200	194
1919	197	142	162	149	209	206
1920	302	170	217	262	318	226
1921	163	143	164	122	179	147
1922	156	140	158	117	132	149
1923	150	112	163	113	125	154
1924	161	102	162	127	149	150
1925	172	106	156	137	170	159
1926	178	112	187	138	170	151
<i>By months</i>						
1926						
January.....	165	99	160	137	183	156
February.....	177	102	169	131	183	155
March.....	176	105	177	135	179	152
April.....	176	109	178	142	173	151
May.....	183	117	198	141	170	152
June.....	188	119	200	136	171	152
July.....	186	119	190	134	169	151
August.....	184	119	196	140	166	149
September.....	182	119	204	143	161	151
October.....	182	114	195	139	161	150
November.....	174	111	119	140	159	148
December.....	163	112	181	139	158	147

tively. The relative cheapness of gasoline is worthy of particular note during a year when crude oil stocks were reduced. The trend of crude oil prices in percentages of the general price level (that is, corrected for variations in the purchasing power of the dollar) is given for the last 14 years in Table 7.

TABLE 7.—*Index Numbers of Oil Prices Expressed in Percentages of the Average Price Level of All Commodities*

Prices of All Commodities = 100

Year	Crude Petroleum	Gasoline	Kerosene	Fuel Oil	Lubricating Oils	All Commodities ^a
1913	100	100	100	100	100	100
1914	85	80	99	87	103	100
1915	65	74	89	67	96	100
1916	92	95	80	80	94	100
1917	88	75	61	83	71	100
1918	100	72	67	98	103	100
1919	96	69	79	72	101	100
1920	133	75	96	116	140	100
1921	111	97	112	83	122	100
1922	105	95	106	79	89	100
1923	98	73	106	74	81	100
1924	107	68	108	85	99	100
1925	108	67	98	86	107	100
1926	118	74	124	91	113	100

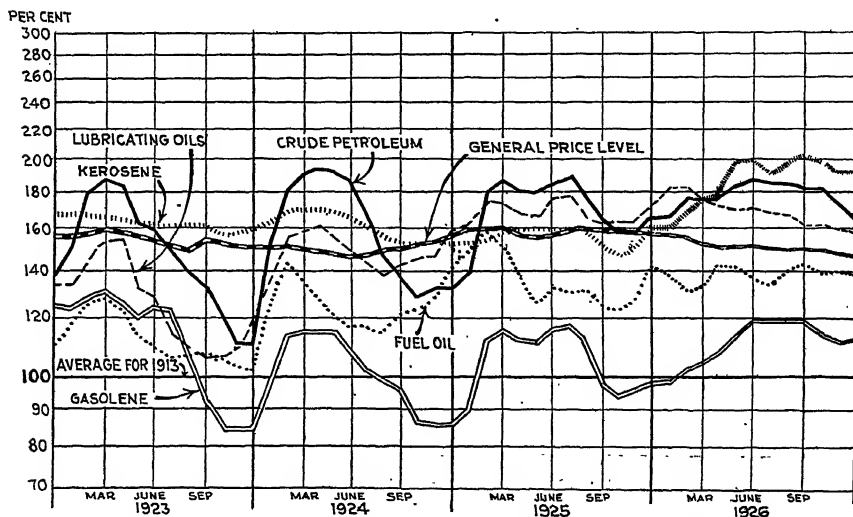
^a U. S. Bureau of Labor Statistics.

FIG. 10.—TREND OF THE RELATIVE PRICES OF CRUDE PETROLEUM AND ITS PRINCIPAL DERIVATIVES IN THE UNITED STATES BY MONTHS, IN PERCENTAGES OF THE AVERAGE PRICES IN 1913 AND IN COMPARISON WITH THE GENERAL PRICE LEVEL. (PRICES IN 1913 = 100.)

SUMMARIZED TREND

As a summary of the major trends that have manifested themselves in the petroleum situation, Fig. 11 is presented. This chart shows for a period of years the demand for gasoline, the domestic production of crude petroleum, and the ratio of gasoline produced to crude oil charged to stills. To each set of data a trend line, or series of trend lines, has been fitted. These trend lines are compound interest curves fitted by the method of least squares; as the data are plotted on semi-logarithmic paper, the trends appear as straight lines. Production has changed its

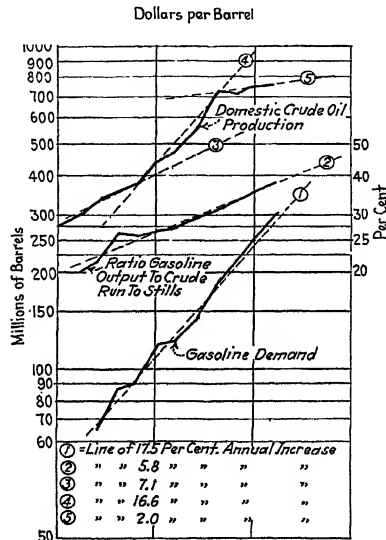


FIG. 11.—DIAGRAMMATIC REPRESENTATION OF THE BASIC ECONOMIC TRENDS IN THE PETROLEUM INDUSTRY, ILLUSTRATING THE BEARING OF THE INCREASING RECOVERY OF GASOLINE UPON THE QUANTITY OF CRUDE OIL REQUIRED TO MEET THE DEMAND FOR GASOLINE. TREND LINES ARE COMPOUND INTEREST CURVES FITTED TO THE DATA BY THE METHOD OF LEAST SQUARES.

trend twice during the period shown; the trends of the other two factors continue through 1926 unaltered. The chart illustrates how the rapid growth in the demand for gasoline has been met by a crude oil supply expanding (with the exception of a brief period, 1920–23) at a substantially slower rate. The present trend of the crude-production and gasoline-demand curves cannot be safely projected into the future; it is probable, however, that the ratio curve for some time ahead will tend to follow its present course, continuing to multiply by a growing factor the economic value of the barrel of crude and relieving the supply of crude oil from the necessity of fully keeping pace with the further expansion in the demand for gasoline.

Subsidence and Earth Movements Caused by Oil Extraction, or by Drilling Oil and Gas Wells*

By W. T. THOM, JR.,† WASHINGTON, D. C.

(New York Meeting, February, 1927)

INTEREST naturally attaches to fissuring and subsidence of the earth's surface, whatever the cause may be, and the induced movement and fissuring of the impervious strata overlying an oil sand is of even greater practical importance because of its bearing upon the recoverability of the portion of the oil not already extracted or extractable by ordinary production methods.

It is probably true that significant ground movement induced by oil production occurs only in oil fields in relatively young formations, where the oil comes from incoherent or only slightly cemented sands and oil-soaked clays, but inasmuch as a great proportion of our "gusher" wells produce from "running" sands and yield a large part of the total oil output, it seems worth while to make an effort to inspire the collection and recording of data indicating the nature and magnitude of oil-field subsidence, both surface and subsurface.

GOOSE CREEK FIELD

The single known instance in which the nature and magnitude of the surface subsidence incident to petroleum withdrawal has been accurately ascertained is that in the Goose Creek oil field of Texas, described by Pratt and Johnson.¹ There subsidence caused the submergence of a considerable area within the oil field, and, because the title to the naturally submerged lands of Texas rests in the State, it became necessary to determine whether the submergence was due to natural or artificial causes. A detailed study revealed that the sunken area conformed very closely to the limits of the productive oil field and that the cumulative depression over an eight-year period ranged from zero at the margin to 3.25 ft. at the center. This depression was accompanied by the development of local marginal surface fractures along which the subsiding area

* Published by permission of the Director of the U. S. Geological Survey.

† Geologist in Charge, Fuel Section, U. S. Geological Survey.

¹ W. E. Pratt and D. W. Johnson: Local Subsidence of the Goose Creek Oil Fields. *Jnl. of Geol.* (Oct.-Nov., 1926) 34, 577-590.

dropped as much as 16 in., the fissuring causing earth shocks of sufficient violence to rattle dishes and spill water in houses near the field, and to disturb street and lawn surfaces and house foundations.

Calculations also show that the volume of the oil, gas, water and sand extracted to date from the Goose Creek field exceeds 500,000,000 cu. ft., which is several times the volume of the induced surface depression. It seems probable therefore that the disruption of the deeper strata must be far greater than that visible near the surface, and that readjustment of the surface to underground conditions will continue after oil production has ceased. Furthermore, it would seem surprising that Goose Creek should present the single, isolated example on the Gulf Coast of the sort of movement described, and it may well be that rechecking of elevations in other fields may reveal a similar slight subsidence.

Ground movements of considerable significance, though of different type, are familiar to petroleum engineers who have been called upon to deal with "wild" oil and gas wells which have "blown out" or developed craters, and it would seem well worth while to gather information about the phenomena accompanying such "blow-outs." In referring to such a well in the Sarvas gas field in Transylvania, Clapp² says:

The original rock pressure at the bottom was 384 lb. per sq. in. Owing to inexperience with gas-field methods, difficulty was experienced in closing the well. A commission headed by Mr. Ferenc Bohm was sent to America to investigate ways and means, and on his return in June, 1910, the well was successfully closed.³

By this time gas was escaping from the earth at many places within a radius of 1000 ft. from the well, having forced its way to the surface through the imperfect packing at the 430-ft. gas horizon, and the laborious method of pumping the space outside of the casing full of "cement milk" was resorted to in order to save the well and the field.

From the time of successfully sealing the sands in July, 1911, until the end of October in the same year the well was closed continually, but on October 26 a severe shock occurred, followed by a second and third shock on the morning of October 29. At 2 a. m. the well guard heard a heavy rumble and at 2:30 o'clock came the severest shock, at which time debris was thrown into the air at a distance of half a mile from the well, and gas commenced to leak in many places from an area of several acres. The shock was felt in an elliptical area about five miles in length along the major axis of the dome.

Wells that have developed similarly widespread surface fissures before being controlled, and others that have "cratered," are familiar features of the oil history of northern Louisiana, southern Arkansas, California and other parts of the United States and of the world.

² F. G. Clapp: Notes on Natural Gas Fields of Transylvania, Rumania. Amer. Assn. Pet. Geol. *Bull.* (1924) 8, 202-211.

³ Ferenc Bohm: A Kissarmasi Gaskut Tomitsse, in A Magyar Merokes Epitez-Egylet Kozlonye. *Erv.* 7, Szamabel, Budapest, 1912.

THE DOS BOCAS CRATER

Probably the greatest crater ever produced by the "blowing out" of an oil well is that which marks the site of the famous Dos Bocas gusher in Mexico. This well, 52 miles south of Tampico, "drilled itself in" at a depth of 1834 ft., and, according to Redwood,⁴ "the casing was hurled out of the hole and huge cracks opening in the earth wide enough for people to walk about in." Oil issuing from these fissures took fire, which was only extinguished when salt water began to issue around the well. Gradually the ground sank or was washed away until a lake of steaming salt water, a half-mile wide, marked the former well site, and a stream of hot salt water, reported by Redwood at 15,000,000 bbl. a day flowed from this crater. That the crater was formed largely by the erosive power of the water is probable, but that a considerable part of the collapsing ground sank into the cavern whence the oil and water came also seems equally probable.

Indications of great subsurface movements resulting from oil and gas extraction are yielded also by the productive histories of the many oil wells that discharge great quantities of sand with the oil. Of these many subsequently require "side-tracking" and redrilling or other treatment, because the clays overlying the oil sand have collapsed into the cavity produced by the outflow of sand, thus blocking the perforations through which the oil and sand had previously entered the well casing. The volume of sand, rock and mud thrown out by "wild" wells is often surprisingly large.

UNCONTROLLED WELLS

In Trinidad, for example, great quantities of sand and rock have been thrown out by uncontrolled wells. One such well, which produced for four days before "sanding up," ejected thousands of tons of sand, which covered the neighboring territory, being 5 ft. deep in some places. In other instances, oil-soaked clays penetrated by a well have flowed into the bore hole and have poured out upon the derrick floor in such volume as to cause the abandonment of the well.

In describing one of the Baku gushers or "fountains," Redwood⁵ says:

Some idea of the mass of matter thrown up from the well could be formed by a glance at the damage done on the south side in 24 hr.—a vast shoal of sand having been formed which buried to the roof some magazines and shops, and had blocked to the height of 6 or 7 ft. all the neighboring derricks within a distance of 50 yd. Some of the sand and oil had been carried by the wind nearly 100 yd. from the fountain.

⁴ Sir Boverton Redwood: *A Treatise on Petroleum*, p. 131. J. B. Lippincott Co., 1922.

⁵ Sir Boverton Redwood: *Op. cit.*, 7.

Doubtless numerous comparable instances of sand discharge can be discovered in Gulf Coast and California oil-field history. Such a study undoubtedly would be valuable in throwing light on subsurface conditions, which affect the recoverability of the dormant oil in old oil fields. The magnitude of the disruptive effects produced underground by oil extraction, and their possible bearing on present and future oil production is suggested by Redwood⁶ in describing the Bibi-Eibat field, where drilling to lower sands is difficult "owing to the upper sands having been much disturbed by fountains in the previous exploitation of the field."

It is much to be hoped that petroleum engineers and geologists who have quantitative and qualitative data on subsidence or related phenomena will bring their information to the attention of the Committee on Ground Movement and Subsidence in order that it may be recorded and made available in assembled form. Supporting data, bearing upon depression due to fluid discharge from cemented sands, may also perhaps, be obtained from those interested in artesian water conditions, as Meinzer suggests⁷ that the withdrawal of artesian water in North Dakota has induced a depression of the surface amounting to several inches.

DISCUSSION

H. G. MOULTON, New York, N. Y.—This paper brings up some very interesting questions regarding the relative rights of the owners and producers of petroleum on the one hand and the owners of the surface on the other, in the event of subsidence of the surface following the extraction of petroleum and the resulting damage. Mr. Bosworth has brought to the attention of the Committee on Ground Movement and Subsidence some of the legal problems connected with the removal of coal beneath surfaces held under ownership separate from that of the mineral, and litigation arising from subsidence resulting from petroleum extraction may involve all of the questions discussed by Mr. Bosworth and additional problems inherent in the nature of the operation.⁸

If the surface subsides as the result of the extraction of petroleum from unconsolidated sands, the extent of subsidence may be very great, first because of the fact that the oil drawn from a well in a given property may travel through the sands from adjacent property, and therefore the removal which causes the subsidence may be at a considerable distance from the well and beneath land not owned or controlled by the operator, and secondly because the angle of draw may extend the subsidence far beyond the area of extraction.

With the development of improved methods through which the percentage of extraction of the petroleum in the sands is increased, the extent and degree of subsidence may increase, and some very difficult legal questions may arise in cases where oil leases contain no waiver of damages as a result of the depression of the surface.

⁶ Sir Boverton Redwood: *op. cit.*, 10.

⁷ O. E. Meinzer and H. A. Hard: The Artesian Water Supply of the Dakota Sandstone in North Dakota, with Special Reference to the Edgeley Quadrangle. U. S. Geol. Survey *Water-Supply Paper* 520-E (1925) 90-93.

⁸ Robert G. Bosworth: What Duty to Support the Surface Does a Subsurface Owner Owe? A. I. M. E. *Pamphlet* No. 1640-A-F.

This is the first time that the matter of subsidence resulting from the extraction of petroleum has been brought to the attention of this committee. The subject may be of more importance in the future and it is possible that important and involved litigation may develop. It would be helpful to the petroleum industry if the legal questions that may be involved could be brought to the attention of operators at this early stage of the development of subsidence problems, and the conflict of legal opinion which has arisen in connection with coal-mining operations might be avoided.

G. S. RICE, Washington, D. C.—While in Great Britain in 1923, I attended a meeting of the North of England Section of the Institution of Mining Engineers and heard a very interesting discussion. It arose from the fact that subsidence of the surface had occurred not far from Newcastle, where it had caused some damage to buildings. The question arose as to the reason. The arguments presented by one side were that, as that part of the country was deficient in water, for long periods water had been pumped from the horizon of limestone dolomite and that the gradual dissolving of the materials in that horizon had brought about the subsidence. The other side said it was due to mining in the vicinity, but the owners of the mine said their operations were not near where the subsidence occurred. The question was not settled then. I have not seen any reference to it since, but it evidently involves some legal phases.

E. L. DEGOLYER, New York, N. Y.—I am very much interested in one phase of the matter touched on by your chairman, both in introducing Dr. Thom's paper and in discussing it. That is the question of fixing the liability. A very careful examination has been made in the Goose Creek case because the question of title became involved. As the result of subsidence, part of the land became submerged, and the state of Texas sued for title to the submerged land. I believe that the question of responsibility for damage by subsidence has not come up yet, but as I remember the paper of Pratt and Johnson, I think they showed that the areas of greatest subsidence were more or less coextensive with the areas of greatest production.

I think it would be certainly a sane method to more or less assess the damage on the various leaseholders according to the amount of liquid—oil and water—they have removed from the subsoil. The only other remark that I have to contribute in this connection is with regard to the Dos Bocas well in Mexico, which was mentioned by Dr. Thom and with which I am quite familiar. I do not think it belongs in the category with Goose Creek. It was strictly a case of cratering. The producing rock is a limestone, a very competent formation, and one quite able to support its overburden after the oil had been removed. I have some surveys of the edge of this crater at progressive intervals of a few days apart for the first three or four months of the life of the well during which the rate of change in the form of crater was greatest. I have also a survey made some five or six years later when the crater was much larger. In this case I think the casing simply blew out, the walls fell in, and the shale was broken up, dissolved and carried out to the near-by lagoon in the form of mud; an operation, by the way, which is still going on.

I do not know that there has been a very steady increase in recovery. I think it has been not so much a constant improvement in recovery as that within the past year or two some new methods have come in. In the first case, the air lift for lifting the fluid out of the hole is much more important than ultimate recovery; that is, the use of air drive whereby air or gas, preferably gas, is pumped into a pressure well and builds up rather slowly a pressure which drives some of the remaining oil in the sand out to bleeder wells on the outside rim.

I also think we are faced with the mining of oil in the not too distant future. As far as I can see it today, it looks as though we shall have to open a mine and drive drifts either above or below the oil sand at a safe distance, thereby putting into the sand

innumerable wells which would be only a few feet in length and thus increasing the recovery tremendously.

That brings up another question: What makes the sand come down? Theoretically, particularly in the hard sand, we are taking the oil out of the pore spaces. If the sand is hard, I believe it is competent generally to hold up the surface, even after the oil has been removed. Go back to the Mexican cases again. There are half a dozen or a dozen fields very similar to Dos Bocas with this limestone as the oil-containing rock, where from 100,000,000 to 150,000,000 bbl. of oil have been removed from the field and where I think there has not been any subsidence, and probably never will be any.

It seems to me that a year or two ago I read in some of the British engineering journals that when the subways were built in London (London, as you know, rests on the Oxford clays, at any rate cursory clays of comparatively unconsolidated sands) they drained the water content out of a number of sands and the foundation suffered considerably from subsidence because of the movement in the sand after the water was out. I suppose something like that happened at Goose Creek after part of the oil and part of the water was out. The sand, which is entirely unconsolidated, responded more readily to the pressure, and in the case of oil I should think the sands probably slid very easily one on the other, as they did when they were originally laid down.

As to the duty of the operator with respect to the surface, there is a provision with respect to surface damage in most leases, but it covers the ordinary operating damage and I do not think the matter of damage by subsidence has ever even occurred to the oil people nor do I believe it is covered in any oil lease.

There has been so little subsidence, and it is regarded as one of those things that is unlikely to happen. It seems to me that it would be well worth attention, especially in areas like the Gulf Coast and some in California, where the sands from producing oil are known to be unconsolidated sands, and it could probably best start there. The consideration of the possibility of that sort of thing would receive more attention from the oil operators themselves.

R. G. BOSWORTH, Denver, Col.—There are questions of subsidence in Michigan salt mines. It is presenting a real problem there. As a matter of fact, it is certain that if you pull something out, the aching void has to be filled in somewhere or you are going to run into the problem of subsidence. From the standpoint of the law, the interesting question thereupon immediately comes up as to relative duties. The same thing applies in all the various branches of mining.

If ever there were a field of law where each case or each class of cases should be determined upon the particular facts involved, it is in this question of subsidence. Very different questions, as Mr. Moulton suggested, will come up with regard to the subsidence due to oil operations because of the different fundamental features of that type of mining as compared, we will say, with coal. The chief thing that all of us should bear in mind would be an interest to see that the courts do not hark back to some ancient precedent which does not fit, and to hold them closely to the facts involved in the particular type so that there shall not be the error in the cases involving subsidence in oil operations which I feel so strongly has crept into the coal cases.

W. T. THOM.—I would like to correct one impression which apparently I unintentionally conveyed; namely, that I supposed that the Dos Bocas cratering was similar in its general character to the subsidence at Goose Creek. I think my views entirely coincide with those of Mr. DeGolyer that cratering is the major feature of the Dos Bocas well. The suggestion he offers, that most of the material was carried away, interests me, because, as I suggested, the oil and water which issues from the Mexican fields in such volume apparently comes from caverns or open spaces of considerable

size in the limestone. The limestone itself supports stress, but in so far as large openings occur, and as one might say act as bins at the bottom of a hopper to receive material sliding down the crater walls, it seems to me likely that much of the material originally occupying the volume of the present crater did sink to the bottom of the openings in the limestone and so actually aided in displacing the water which has issued from the crater.

In addition to the other thoughts brought up here, and in addition to the legal problems presented by subsidence, there are closely related problems of production engineering which Mr. DeGolyer has suggested. In a number of fields attempts are being made to restore the pressures originally existing in the oil sands as aids to the recovery of the remaining oil, and the possibility of such restoration may depend upon whether or not our original fluid extraction has so ruptured the impervious beds above the oil sand that our reservoir is no longer air-tight.

I believe engineering thought should turn to devising ways and means for maintaining as long as possible the original pressures in the sand and for allowing the diminution of those pressures as gradually and as extensively as possible so that there will never be great differentials of pressure and, therefore, great differentials of movement between adjacent areas. If we can succeed in lowering the whole surface over an oil field slowly and uniformly, the danger of dangerous disruption of the surface will be minimized, and also the danger of rupturing the sealing strata. It is greatly to be hoped that the engineers and geologists of other companies will follow Dr. Pratt's example and establish gages and measures and marks by which they can estimate the nature and degree of any subsidence that may now be in progress in our oil fields.

W. E. PRATT, Houston, Tex. (written discussion).—My paper in the *Journal of Geology*⁹ on the Goose Creek subsidence describes the most important subsidence of the surface in oil fields with which I am acquainted. I have had established a series of tests on other fields which are designed to reveal any subsidence of the surface, but I have no positive results on these observations as yet.

The only other case of surface subsidence over an oil field that I can substantiate is the case of the Jennings field in Louisiana. This subsidence did not come under my direct observation, since it took place soon after the early development at Jennings. However, I have affidavits and detailed statements from the late William Kennedy, a competent observer, as to the facts in this case. These statements reveal that the surface subsided over an area of several acres, immediately overlying the salt dome that marks this field. It took place following the peak of development, and it was made evident by the fact that a lake formed in the depression which resulted from the subsidence.

Of course, the Jennings subsidence may be a little different in character from the Goose Creek subsidence, since the Jennings field has a known salt dome not too deeply buried, whereas in the Goose Creek field the deepest hole has never penetrated any salt. The subsidence at Jennings, therefore, might possibly be caused by the solution of the upper part of the salt. Mr. Thom has mentioned the subsidence over domes from which sulfur has been produced. This is also universal, and in the same category may be placed the subsidence over domes from which salt has been produced by circulating brine, such as the well known subsidence over the Palestine dome in Anderson County.

R. C. PATTERSON, Washington, D. C. (written discussion).—There is considerable evidence that in California there is subsurface subsidence in the oil sands or strata generally, under certain conditions, against which proper precautions have to be taken; but in this part of the country there has been no surface subsidence that is known to have been caused directly by the extraction of oil.

⁹ W. E. Pratt and D. W. Johnson: *Op. cit.*

The larger operators in the Midway fields do take the precaution of placing cement foundations under the entire derrick to prevent settling of the surface formations. The surface formation sloughs very rapidly when subjected to the quantity of water used for drilling purposes about a well. The greatest precautions are necessary in the Midway-Sunset area, where water is known to settle the surface formation anywhere from 4 to 5 ft. to 20 to 25 ft., depending on the volume of water and length of time.

The surface formation of this area, which is no doubt part of an old lake-bed fill, has in a number of places been logged as deep as 600 ft. While such surface subsidence has to be given consideration by the operator, it can not be attributed to the extraction of oil.

A very interesting example of subsurface subsidence caused by sand extraction was afforded by a company operating in the Midway oil field. This company has drilled 33 wells on 126 acres, the average depth per well being 1300 ft. (average thickness of the sand, 300 ft.). The oil sand in this acreage is estimated to average 13,068,000 cu. ft. per acre.

This property has been producing for about four years—irregularly, on account of market conditions—and up to Jan. 1 of this year (1927), 379,471 bbl. of oil has been recovered, together with 26,055 cu. yd. of sand pumped into the sumps. Of this quantity, 7020 cu. yd. of sand was produced from one well alone, which was far above the average figure.

On an adjoining section, a well was completed on Dec. 14, 1923, and produced until March 20, 1925. About this time the casing collapsed in the hole and it was necessary to redrill the well. In redrilling, a cavity was encountered at the top of the oil sand; and in order to prevent further caving at the top of the oil sand it was necessary to fill the cave in the well with 50 tons of rock, cement, sacking materials, and four pieces of 8 by 8 timbers, which totaled 32 ft. in length. It is assumed that the 8 by 8 timbers acted as a reinforcement and prevented the concrete aggregate from sloughing off into the sands. In this manner the redrilling of the well was finally completed, and the well put to producing again.

This occurrence shows that large cavities exist in oil-bearing formations, although they may not appear at the surface; caused, no doubt, as in this case, by the removal of the sand incidental to the recovery of oil.

On government leases operated by another company, 33 wells have been drilled and put to producing within a period of four years. Some of these wells had an initial production as high as 6000 bbl. of oil per day. The gas volume was rapidly exhausted, and during the transition period between that of the flowing well to that of a pumping well, the sands, when the gas pressure was reduced, evidently started a subsurface movement. This movement caused the casing to collapse in the oil-bearing strata. The repair work necessary to remedy these defects has caused the company an enormous expense. To date it has been necessary to redrill the oil strings in seven of the 33 wells. Adjoining operators have found it necessary to make similar repairs on many of their producing oil wells.

In the Coalinga fields it has been necessary to repair many wells in which the casings collapsed in the oil-producing formation. This field, one of the oldest in the State, has had a long producing life. A cave in this field is usually preceded by the production of a muddy slush, which in turn is followed by a cave of sand and shale, which causes the casing to collapse. The damaged casing left in the well usually falls off to the side far enough to give clearance for redrilling. After redrilling and setting oil strings, the companies have found it most economical to set blank pipe opposite the previous known cave depth, and to pump in sufficient cement through perforations to reinforce the oil string and produce the lower strata of oil sands, which do not cave in so readily.

742 SUBSIDENCE AND EARTH MOVEMENTS CAUSED BY OIL EXTRACTION

In southern California (the Brea Canyon area), the oil fields are located along what is known as the "Whittier Fault." In 1914 or 1915 there was an earthquake, which is known to have caused subsurface movement in the oil-producing horizon in this area, and during which the casing in six wells collapsed. The peculiar part was that the greater number of the casings collapsed in the oil-bearing horizon, which would indicate that there was considerable slippage in the horizon that had produced large quantities of oil.

Magnesite Mining in California

By LEROY A. PALMER,* SAN FRANCISCO, CAL.

(New York Meeting, February, 1927)

ALL the domestic production of magnesite during 1925 came from two states, California and Washington. Of a total of 120,660 tons of crude ore, 64,600 tons, or 54 per cent., were produced in California.¹ Of this quantity California produced about 99 per cent. of the caustic calcined, but only 10 per cent. of the dead-burned. The largest producer was C. S. Maltby, of San Francisco, who operated the Red Mountain mine, in Santa Clara County, and the Sampson mine, in San Benito County, under lease. These two mines have been studied by the author, and offer interesting comparisons as to mining and treatment methods and final products.

GEOLOGY

The general geology of the magnesite deposits of California has been described by Hess,² Bain³ and others,⁴ and although a detailed discussion is not necessary here, a brief summary will assist in understanding mining conditions.

California magnesite, when comparatively pure, is usually a white, fine-grained rock, with a conchoidal fracture, occurring in veins and lense-like masses in serpentine. The veins vary considerably in width, and usually occur in a series of more or less parallel veins intersected by other series or systems of veins, probably following original or secondary fissures in the enclosing serpentine. The most generally accepted theory of origin is that the magnesite was formed by the action of surface waters charged with CO₂ percolating downward through fissures and changing the magnesium silicates of the serpentine into magnesium carbonate.

* Mining engineer.

¹ J. M. Hill: Magnesium and its Compounds in 1925. Mineral Resources of the U. S. Bur. Mines (1926).

² F. L. Hess: Magnesite Deposits of California. *Bull.* 355, U. S. Geol. Surv. (1908) 17.

³ G. W. Bain: Types of Magnesite Deposits and Their Origin. *Economic Geology* (Aug., 1924) 412.

⁴ See bibliography on magnesite by R. B. Ladoo. *Non-Metallic Minerals* (1925) 341.

COAST RANGE DEPOSITS

The writer's observations have been chiefly in the Coast Range of California, in which his closest studies have been of the deposits at Sampson Mountain, in San Benito County, and at Red Mountain, in Santa Clara County.

Operations at the Sampson deposit have been in a quarry which has opened a wide vein to shallow depth only, so that no great opportunity is offered for study, but at Red Mountain a number of veins have been opened to a depth of several hundred feet, so that conditions are favorable for observing their characteristics. These observations indicate that the veins of the Coast Range are typical of those occurring in serpentine.

The Coast Range has a general northwest-southeast trend. The most widespread formation is the Franciscan, the principal member of which is a massive greenish-grey sandstone which has been intruded by a number of large dikes and batholithic masses of ferro-magnesian rocks; pyroxenites, and peridotites. The magnesite is found in these basic intrusives, generally where they have been altered to serpentine.

Naturally, the intrusions were accompanied and followed by extensive faulting and fracturing, which opened many channels for the circulation of the CO₂ solutions, and the veins thus formed frequently have at least one well-defined wall, clean-cut and slickensided. This is particularly the case where the crushing has been on only one side of a major fault plane, so that the solutions have had easy access to a crushed zone that has led them away from the solid wall which was less susceptible to attack and chemical reaction.

MOVEMENT OF OREBODIES AFTER FORMATION

The orebodies give ample evidence that considerable movement took place after they were formed. This movement was both lateral and longitudinal. It shows longitudinally in numerous strike faults that usually occur on the hanging-wall side of the vein. Dip faulting is not a common phenomenon, but there are numerous instances of lateral pressure in many distortions of the orebodies. The most noticeable of these is in the White Queen vein where the fracture zone and the orebody have been forced into a huge crescent having a large bulge in the middle and tapering ends.

The strike faults are usually on the hanging-wall side of the veins, and have resulted in smearing over the wall a layer of ore, varying in thickness from a few inches to 2 or 3 ft. This layer of ore acts as a protective seal of the wall, and prevents oxidation after the deposit has been exposed to the atmosphere by mining. It operates so usefully in this respect that, when its thickness is not so great as to represent a substantial loss, it is found advisable in mining to leave it, as the

checking of the oxidation of the wall is a material assistance in supporting excavation.

Although general conditions favor the conclusion that the magnesite was formed by percolating water, the Red Mountain deposit gives interesting and indisputable confirmation. Open fissures have been found in which one wall bears a crust of magnesite like the right-hand lump in Fig. 1, the appearance of which leaves no doubt in the observer's mind that it was formed by water trickling over the wall. In fact, had the water dropped from the roof, instead of flowing over the wall, it would have resulted in a stalactitic growth instead of the crust observed. Even further confirmation is found in the fact that thin films of a similar nature have formed recently in some of the worked out and abandoned stopes.

Although there seems to be no reason to believe that the deposits at Red Mountain and Sampson Mountain are dissimilar in their origin, as



FIG. 1.—SPECIMENS OF MAGNESITE.

both occur in serpentine, with like appearance on the surface, the Sampson deposit has not been developed to an extent that permits a satisfactory comparison of the two.

Mining

The deposits in the vicinity of Red Mountain occur along fracture zones having a trend within a few degrees, either way, of north and south. The major fractures strike somewhat east of north, and are crossed by minor fractures that intersect them at acute angles. The dips vary widely, but average from 45 to 50°, and are generally in a westerly direction, although occasional easterly dips occur.

The veins vary greatly in width. The stopes are from 6 to 15 ft. wide, with an average of 10 ft. Extreme widths reach 40 ft., but this is rare. On the other hand, the mineralized area may narrow down to a mere stringer. When such contractions in the vein are encountered in horizontal operations, they are followed out, and it is a frequent occurrence to find another swell with a body of ore of sufficient width to mine profitably. Where such contractions are encountered in sinking, the indications are that they mark the lower limit of the orebody, thus adding support to the theory of downward solutions, although the deposit has not been followed to sufficient depth to prove this statement conclusively.

The Red Mountain deposit has been opened up over an area of several thousand feet in both length and width. A large number of veins have been discovered and considerable exploration work has demonstrated the existence of a deposit of unusual size. Knowledge of the actual extent is still limited, however, as almost the entire surface is covered by a growth of brush which is literally impenetrable. This is not causing any great concern at present, as the ore reserves are ample for the time being; but, when occasion arises, the brush will be burned and the surface given a thorough prospecting.

Red Mountain Mines

The ore from the Red Mountain mine has been shipped from four groups of workings or mines, and three camps have been maintained. More recently, operations have been concentrated at the White Queen and White Diamond mines and carried on from one camp near the center of the property. Transportation from each mine is by aerial tramways,



FIG. 2.—MAGNESITE CALCINING PLANT AT RED MOUNTAIN.

the two converging and dumping to the same bin at the calcining plant (see Fig. 2).

A quarry was opened on the White Queen, on the crescent-shaped deposit mentioned, and a large tonnage was extracted before conditions were changed so as to make underground work more economical. Levels have been driven beneath this deposit and raises extended to it, and it is now worked by stoping. There are two levels, 100 ft. apart, but the principal production is from the White Diamond group.

At the time these notes were made one shoot on the White Diamond vein had been worked horizontally for 700 ft., through widths ranging from 3 to 20 ft., with an average of 10 ft. The shoot on the Smith vein, followed for more than 400 ft., has an average width of 6 ft. The Ollalla has been opened to a depth of 225 ft., with a length of 200 ft. on each level, the width ranging from 4 to 24 ft. and averaging 12 ft. The Ehrmann had an average width of 15 ft., through a vertical range of 200 ft., but development on this, the Cummings, and the Silver, has been of more

limited extent than on the others. These figures are given somewhat in detail in order to convey some idea of the extent of the deposit.

The steep mountainside makes a condition that is ideal for operating through tunnels. The Diamond group has been opened for a range of 500 ft. through four levels. A main haulage way was driven and operated for several years until it was demonstrated that the ore extended for a considerable depth beneath it, and then the present haulage level was driven 175 ft. below the old one.

Method of Opening a Deposit

In opening a deposit, a main entry is driven in the footwall parallel to the vein. These entries are 7 by 7 ft. in cross-section and require little timbering, as the unaltered serpentine is not materially affected by atmospheric agencies. At intervals of 150 to 200 ft. main raises are driven, with a cross-section of 5 by 7 ft. to the next level. These raises are carried in the footwall until they meet the vein, and this is then followed upward. Between the main raises, at intervals of 30 ft., secondary raises are carried up approximately 15 ft., at which level they have penetrated a few feet into the vein. They are then turned each way at angles of 45° and carried in the vein on this slope until they meet.

Stoping is then done by drilling holes in the back, and blasting down and drawing off only such ore as necessary to give working room. Occasional pillars are left to support the walls, but, with this exception the method is typical shrinkage stoping, and need not be described. Operations cease with a back from 6 to 8 ft. thick between the stope and the next level. This ore and the stope pillars will be removed when the time comes to abandon that part of the mine. Stoping is carried on by working upward from the main levels, driving a level in the vein to the end of the ore shoot and working backward until the ore is removed. Fig. 3 illustrates one of the stopes in the White Diamond mine.

Impurities

No mechanical treatment of magnesite ore for improving its grade is practiced at the California mines, and consequently the impurities allowed in mining are restricted to narrow limits. Hand sorting is the only method of bettering, and consequently it is essential to break the ore in the stopes as cleanly as possible, and with no more admixture of wall rock than can be avoided. Furthermore, as has been noted, there is frequently a "gouge" along the hanging wall, which latter has been more or less shattered by the processes incident to the forming of the veins. This gouge acts much as a heavy layer of cement, and helps to keep the wall in place so that it is desirable to leave it undisturbed. This also calls for careful mining. For these reasons the tonnage mined per man does not approach that usually attained in shrinkage stoping in metallic

minerals, which is generally carried on in low-grade deposits where a large tonnage is sought with the expectation that any lack of care in mining will be remedied in the mill. At the Diamond the gross output per man is from 3 to 5 tons, but when waste is sorted out and fines screened only 1 to 2 tons reach the kilns.

Main drifts, which have a cross-section of 7 by 7 ft., are driven with two miners and two muckers per shift. The men are paid \$5.50 per shift,



FIG. 3.—SHRINKAGE STOPE IN THE WHITE DIAMOND MAGNESITE MINE.

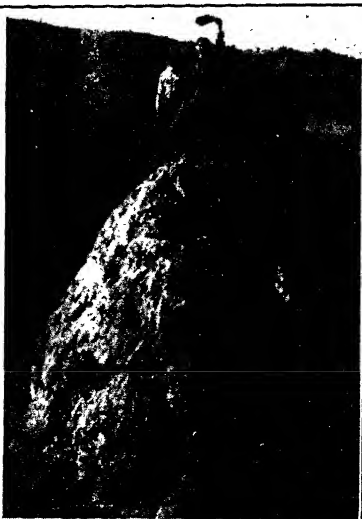


FIG. 4.—MAGNESITE OUTCROP AT SAMPSON MINE.

with a bonus of \$1 per foot for every foot more than five broken per day. A standard center-cut round of from 12 to 15 holes is used, with about 50 lb. of powder per round. From 5 to 6 ft. per round is the usual progress. These drifts cost about \$6 a foot.

The raises are 5 by 7 ft. in cross-section, and cost \$4 a foot. Stopping is done by contract. The company furnishes the machines and air, sharpens the steel, and trams the ore; the contractor furnishes labor and powder only.

The ore is loaded through hand-operated sector gates to 2-ton side-dump cars hauled by a gasoline locomotive with a Ford automobile engine. The various parts of the locomotive were hauled up the steep road to the mine and assembled there.

Ventilation

The loading chutes are only a few hundred feet from the mouth of the haulage-way, and ventilation conditions are such that there is always a

strong outcast from the haulage level, so that the gasoline locomotive can be used with safety. Haulage is through a level driven for that purpose at an elevation of 175 ft. less than the lowest operating level.

At the mine bins the ore is dumped over a 1½-in. grizzly, the undersize from which passes to a ¾-in. screen. This gives three products, lump for the stack kilns, fine ore for the rotary kiln and dust. The latter is wasted, as it is found to run high in impurities.

Sampson Mountain

The surface of the claims at Sampson Mountain is covered by dense brush, characteristic of the Coast Range, so that, as at Red Mountain, it is impossible to obtain an adequate idea of the outcrops or of the extent of the deposit from a study of the surface. Here and there outcrops, some of them of unusual prominence, as illustrated in Fig. 4, indicate that the deposit is of great extent.

Production at Sampson Mountain is from a quarry on one of the large outcrops which has been opened by two benches, one 45 and the other 90 ft. above the haulage level. Each bench is reached by a raise, 4 by 4 ft. in cross-section, driven at an angle of 75° and lagged both inside and outside. The haulage level is 5 by 7 ft. in the clear, well timbered, and 250 ft. long to the raise to the upper bench, the raise to the lower being at the 83-ft. point.

Fig. 5 shows the surface workings at the White Queen mine. Fig. 6 shows the quarry on the upper bench of the Sampson mine. The benches are worked independently of each other as quarries and not as glory holes. Faces are carried to a maximum height of approximately 30 ft. Mining is done by jack-hammers which receive air from an 8 by 6-in. Rix-Gardner compressor belted to a gas engine from a Kelly truck. All holes are vertical or nearly so, varying, according to conditions, from 5 to 15 ft. in depth, with a burden of 5 or 6 ft. The powder used is 40 per cent. gelatine.

Sorting

The raise from the haulage level holes out into the quarry, and tracks radiate from it to the various faces. As the ore is broken down the larger pieces are blockholed, and it is sorted before being loaded into the cars. Color and weight are no guides to the sorters, who judge by the appearance of the fractured surfaces. A clean conchoidal fracture is an indication of good ore; an appearance of stringers or fibers on the surface indicates high silica and a rough, hackly surface lime.

Mining is done by contract, 15 men being employed for an output of 120 tons daily. Two of these men are miners, two are trammers, and the rest are sorters and muckers. The company furnishes the drills and sharpens the steel.

From the chutes the ore is trammed to a 100-ton bin, 150 ft. from the mouth of the tunnel. At the bin it is dumped over a $\frac{3}{4}$ -in. grizzly where a rejection is made of the undersize, as it is found that the fines of the first breaking are high in lime and silica.

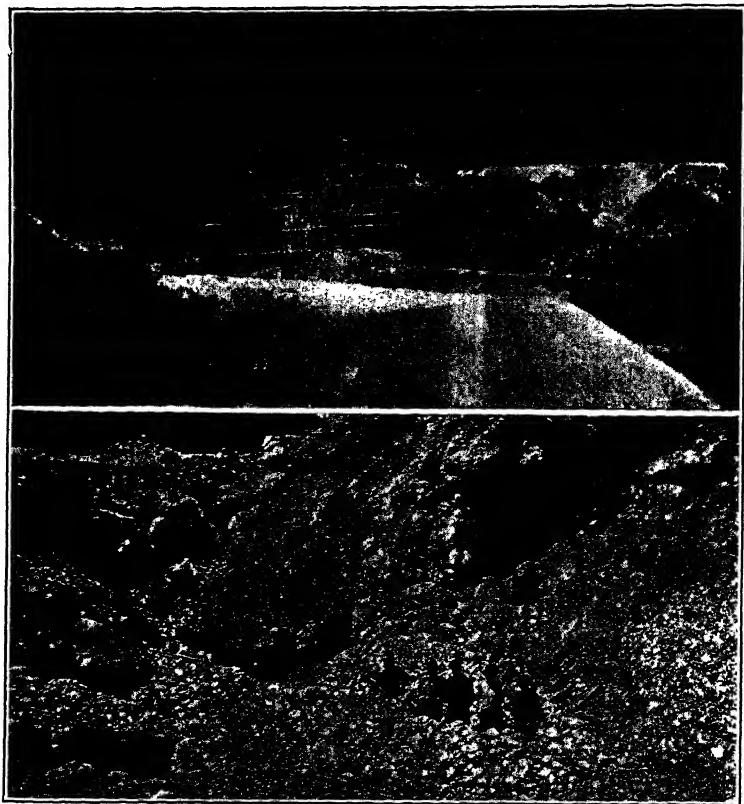


FIG. 5.—SURFACE WORKINGS AT WHITE QUEEN MINE, WITH QUARRY IN THE BACKGROUND.

FIG. 6.—QUARRY ON THE UPPER BENCH OF THE SAMPSON MINE.

TRANSPORTATION

At each locality the mine is well up the side of a steep mountain at an elevation considerably higher than the treatment plant, a condition making the aerial tram an ideal method of transportation.

There are two tramways at Red Mountain. At the White Queen a standard type of 2-rope aerial tram loads from a 200-ton bin and delivers at the kiln, 2600 ft. away, after a drop of 600 ft. At the White Diamond

there is a 100-ton bin serving a home-made "jig-back," having a drop of 425 ft. in 1250, which is giving excellent satisfaction. Both tramways dump automatically at the kilns into a 150-ton bin having one compartment for lump ore for the stack kilns and one for fine ore for the rotary kiln.

At the Sampson, a standard type of 2-rope tramway with $\frac{1}{2}$ -ton buckets transports the ore 4500 ft. horizontally, in which distance it is dropped 1600 ft. and is dumped by hand into a 200-ton bin at the kiln. All ore at the Sampson is treated in a rotary kiln, so that all coarse pieces must be reduced prior to treatment. From the terminal bin the ore is fed without sizing to a 10 by 16-in. Hendy Blake crusher set to 2 in. A belt elevator raises the crushed product 24 ft. and delivers it to a 3 by 7-ft. trommel with 1-in. punched holes, the oversize of which goes to a No. 2 Tel-smith gyratory crusher set to $\frac{1}{2}$ in. Crusher product and trommel undersize go to the 80-ton bin from which the kiln is fed.

BURNING

Magnesite is marketed in three grades: crude, dead-burned, and caustic calcined. The crude or untreated ore is used for various purposes, such as the manufacture of metallic magnesium, magnesia salts, paint, etc. Very little of the California product is marketed in the crude state. Dead-burned magnesite, which is the completely calcined product, has its principal use as a refractory. It should be free from CO_2 and carry not more than $2\frac{1}{2}$ per cent. of CaO and 5 per cent. of SiO_2 , and should contain from 7 to 10 per cent. of iron oxide. The caustic calcine—the "calcine" of the furnaceman—is the light-burned product, and is used for many purposes, such as plaster, cement, wallboard, pipe covering, sound insulation, etc. It has the same lime and silica content as the dead-burned, but should contain not more than 6 to $6\frac{1}{2}$ per cent. of iron oxide. It is customary to have 3 to $3\frac{1}{2}$ per cent. CO_2 but up to 5 per cent. is permissible for most uses.

Coarse Treatment

At Red Mountain the coarse ore is burned in four-stack kilns and the fine ore in a rotary. The stacks are situated so as to be convenient to the coarse-ore bin, one being charged directly from the bin and the others by using short trams. Two sizes of kilns are in use, 6 ft. inside diameter with a capacity of 12 tons per 24 hr., and 7 ft. diameter with a capacity of 14 tons. They are oil-fired, with fire-boxes built around the base, and discharge to individual stacks, $2\frac{1}{2}$ by 30 ft. A charge column of 18 ft. is carried, and the temperature maintained is from 1800° to 2000° F., under which conditions a period of 8 hr. is required to produce the caustic calcined product.

Fines

At the time these notes were made the fines at Red Mountain were being treated in a Scott furnace, which has since been supplanted by a

rotary kiln. As the rotary treatment is described in some detail in connection with the Sampson, only the former method of treatment in the Scott furnace is considered here.

The oversize from the trommel at the mine, $1\frac{1}{2}$ to $\frac{3}{4}$ in., is dumped to separate compartments in the bin at the kilns. From these bins it is fed through a hand-operated gate to a conveyor, 125 ft. long with a rise of 25 ft., which also serves as a picking belt. The serpentine and other impurities are picked out and the ore is discharged to a 50-ton bin, from which it is loaded to cars and trammed to the Scott furnace.

The Scott furnace was designed originally for treating quicksilver ores, and this particular furnace was moved in from a near-by quicksilver mine. It is rectangular in cross-section, 16 by 24 ft., 33 ft. high, and equipped with an oil fire-box at each end. On each side of the furnace there is a series of inclined shelves pitching in opposite directions, so that the ore slides across one shelf and falls from the bottom of that shelf to the top of the one below it and thus slowly runs or "trickles" from top to bottom of the furnace, during which passage it is subjected to the heated gases from the fire-boxes passing through the stack.



FIG. 7.—ROTARY KILN AT THE SAMPSON MINE.



FIG. 8.—TRAMWAY BINS, CRUSHING PLANT, AND STACKS OF THE ROTARY KILN, AT THE SAMPSON MINE.

Rotary Kiln

The Scott furnace has been successful in treating quicksilver ores, and has made a satisfactory product in treating the magnesite, but was discarded in favor of the rotary kiln because of the greater capacity of the latter.

At the Sampson mine all treatment is in a rotary kiln which makes either dead-burned magnesite or caustic calcine, according to the market demand. The kiln, Fig. 7, is 80 ft. long, and is set with a 16 to 1 pitch of 5 ft. The outside diameter is 7 ft. but this is reduced to 6 ft. inside by the fire-brick lining, and further tapered at the stack end to 4 ft. for the

purpose of reflecting the heat back toward the fire-box. An eccentric feeder delivers the crushed ore from the bin to a chain elevator which raises it 12 ft. and discharges it to a water-jacketed feed pipe to the kiln. Fig. 8 shows the tramway bins, crushing plant, fine-ore bin, and stacks of the rotary kiln.

Grading

The grade of the product—that is whether dead-burned or caustic—is governed by the heat in the kiln. For dead-burning this is 2800° F. in the fire zone, which is from the 10 to the 20-ft. point, reckoning from the fire-box end. With this temperature in the fire zone the stack gases are at about 1000° F. In dead-burning, the kiln is given an extra lining in the fire zone by fusing 3 or 4 in. of magnesite dust onto the lining under high heat for about 24 hr. At the time of the author's visit about 4000 tons of dead-burned magnesite had been produced with the lining then in use, and it was still in good condition.

As the Sampson ores are deficient in iron, which is necessary in the dead-burned product, it is added in the form of mill scale. This is stored in a small bin beneath the ore bin and discharged by an eccentrically-operated feeder to the chain elevator that carries the ore to the kiln.

Caustic calcine is produced in the same manner as the dead-burned, but at a lower temperature, 2200° F. in the fire zone and 400° F. in the stack. With this more moderate temperature the reinforcement of the lining is not necessary.

The kiln makes a revolution in 80 sec., and has a capacity of 35 to 45 tons of dead-burned per 24 hr. according to whether the product is burned to a light brown or a black. The analysis of the two is the same, except for a difference in ignition loss, but the black commands a somewhat higher market price. The capacity for caustic calcine is higher than for dead-burned, as the kiln can be fed more heavily and produce 60 tons per 24 hours.

An interesting feature of the process is the fact that the change in the character of the product appears to come suddenly as it passes through the kiln. When a kiln that has been making dead-burned is shut down and cooled, the first 30 ft. (approximately) are found to contain dead-burned magnesite, after which there are 10 ft. of over-burned caustic, then 10 to 15 ft. of caustic, and the remainder is raw ore. These grades do not merge; there is a sharp line of demarcation between each.

FUEL

The kiln is oil-fired, and consumes 60 gal. of fuel oil per ton of caustic and 70 gal. per ton of dead-burned. It has two stacks, 30 in. by 50 ft., which give a natural draft of 4 in. of water. Between the kiln and the

stacks is a dust chamber, 10 by 16 ft., 14 ft. high, with a baffle wall. This chamber settles out the fines from the kiln and, as they are too high in lime and silica to justify recovery, they are wasted.

Labor consists of one man at each tramway terminal, one crusherman, one blacksmith and general utility man on day shift, and two men at the rotary kiln on both day and night shifts.

HANDLING KILN PRODUCT

The product of the Scott kiln is raked onto a hearth, where it cools and from which it is trammed to a chain elevator to the shipping bin.

The product of the stack kilns, which make only caustic calcine, is raked out onto concrete floors and spread in heaps 1 ft. thick for cooling. When cooled it is shoveled into wheelbarrows and dumped into hoppers feeding an inclined conveyor, 205 ft. long, with a rise of 15 ft. This conveyor is used as a picking belt. One man, stationed at the feed hopper, picks out the serpentine and throws it to the waste bin. Three or four other pickers, stationed along the belt, pick out the pieces that are good magnesite as far as composition is concerned but are off color and consequently not adapted to the many uses that require the pure white material.

In calcining, many of the larger lumps do not "cook" through. The process of calcining causes the thoroughly treated ore to take the form of needle-shaped crystals in radiated groups. These groups crumble very easily, the individual crystals falling apart. When a piece of ore does not cook thoroughly the result is an egg-shaped core surrounded by crystals of calcine. Obviously, a separation of the partly raw ore and the calcine is necessary, and this is effected very simply. The picking belt dumps to a 3 by 7-ft. trommel with 1-in. holes, and the tumbling about in the trommel breaks off the crystals; these pass through the holes and the partly raw cores are discharged from the end as oversize and returned to the kilns. One of these pieces of partly cooked ore is shown in the middle of Fig. 1, and at the left is a small pile of the thoroughly cooked needle-like crystals which have been rubbed from the center chunk. The tonnage ratio of the original charge to the kilns to the final product in the bins is about $2\frac{1}{2}$: 1, with a consumption of 1 bbl. of oil to a ton of calcine.

As all the kiln output at the Sampson is finished product, it goes directly to the shipping bins without further treatment. A bucket-and-chain elevator, running at an angle of 45° , discharges it directly to the dead-burned bin or to a screw conveyor which carries it to the calcine bin. The dead-burned bins are of timber frame with a lining of two thicknesses of fire-brick laid with an air-course between to facilitate cooling. The caustic bins are steel-lined and have steel hopper bottoms.

POWER

The power requirements at both mines are small. At Red Mountain three Chicago compressors, each with a capacity of 300 cu. ft. per min., are connected straight-line to 9½ by 14-in. semi-Diesel engines, and furnish air for the drills and for a water pump to the mine. One 60-hp. boiler supplies steam for spraying oil to the kilns and for four small (5 to 10-hp.) engines which operate the elevating and conveying machinery and oil pumps.

At the Sampson a 65-hp. return tubular boiler furnishes steam for the oil burners and for two engines, one 75-hp. which operates the kiln, crushers, elevators at crude bins, and electric light generator, and one 20-hp. for the elevator for the finished product and the screw conveyor at the calcine bin. As previously noted, the compressor at the mine is run by a gasoline engine.

FREIGHTING

From the Red Mountain mine there is a 32-mile haul to Livermore, on the Western Pacific Railroad and one of the main lines of the Southern Pacific Railroad. The product from the Sampson is freighted 45 miles to Mendota on another of the main lines of the Southern Pacific Railroad. Haulage is by motor trucks.

The mountains of the Coast Range are cut by many steep cañons, so that the roads are frequently narrow and tortuous, although the grades are not bad. Where sandstone predominates in the formation there is a good natural roadbed and cuts stand well; but where shale lenses come into the sandstone, as occurs frequently, road trouble is very apt to be encountered, both in hauling and in maintenance. Although freighting does not present any really serious obstacles, it is not entirely easy.

ACKNOWLEDGMENTS

Acknowledgment for assistance in collecting these notes is due to C. S. Maltby, lessee of the two mines, Ralph H. Moore, one of the owners of the Sampson mine, E. Freitag, superintendent of the Sampson mine and Roy H. Clarke, superintendent of the Red Mountain mine.

DISCUSSION

O. BOWLES, New Brunswick, N. J.—I wish that some geologist would tell us a little more than we now know about the origin of magnesite. I am not questioning the origin that is given here, which is borne out by evidence that appears to be correct; but if percolating waters acting upon serpentine in California and Washington have converted these serpentines into magnesite, I wish we knew why none of the eastern serpentines have given us any magnesites. These occur to some extent in eastern Canada, but we have none whatever of any commercial importance except in Washington and California.

E. F. BURCHARD, Washington, D. C.—The same thing is true for the serpentines of Cuba. I found very little magnesite except in extremely small veinlets.

J. MELHASE, San Francisco, Calif. (written discussion).—Palmer's paper does not sufficiently cover the subject as predicated by the title. To one not acquainted with the facts, the impression may be gained that the Red Mountain and Sampson mines were the sole producers of magnesite in the state; whereas magnesite deposits are known in 19 counties and have been worked in 15. The principal production has been from Tulare County, but other deposits occur throughout a belt extending from Mendocino County on the north to San Diego County on the south, a distance of several hundred miles.

The deposits are not confined to the Coast Ranges alone; they occur also in the Sierra Nevada Mountains and in the desert ranges of San Bernardino County. Although these deposits occur generally in serpentine or in altered basic intrusive rocks, there is another type of deposits in which the magnesite occurs interbedded with sedimentary rocks. Notable examples of this type are found at Bissell, Kern County, where the ore occurs in the Rosamond beds, and again near Afton, San Bernardino County, where a large deposit of magnesite lies between strata of conglomerate and sandstone. At neither of these localities is there any serpentine or basic intrusive present.

The bibliography should also include the most recent and comprehensive of all publications on magnesite in California,⁵ which was published after the bibliography of Ladoo, to which reference is made.

⁵ W. W. Bradley: Magnesite in California. California State Mining Bureau, *Bull.* 79 (1925).

Mining Districts and Their Relation to Structural Geology

BY J. J. BEESON, SALT LAKE CITY, UTAH

(Salt Lake City Meeting, September, 1925)

FOR the past fifty years or more, the structural features of the Cordilleran mountain system of western United States have presented some most interesting problems. Any geologist or engineer living in this region, or even making occasional examinations of the mines, will lose much of interest and importance if he is not familiar with the literature. Some of the greatest works on geology have been the result of studies carried on in sections of this region. Among the more important of the earlier works are those of Dutton,¹ King,² Gilbert³ and Powell;⁴ of the more recent Lindgren,⁵ Ransome,⁶ Spurr,⁷ Butler⁸ and others have contributed much. Many of the papers are largely of scientific interest but even these lay the foundation for the commercial application of the broader features of the geology.

I have read many of these works with the keenest interest and pleasure; in fact, some have been read until some of the more important findings seem to be my own. In this paper, however, it is my aim to accept credit for little more than the labor of assembling some of the ideas of others. This is done with the hope that an interesting and valuable discussion will be the result. If some of the ideas presented should

¹ C. E. Dutton: High Plateaus of Utah, in U. S. Geog. and Geol. Survey of the Rocky Mountain Region, by J. W. Powell (1880).

Tertiary History of the Grand Canyon District. U. S. Geol. Surv. *Mon.* 2 (1882).

² Clarence King: Systematic Geology. U. S. Geol. Explor. (1878) 1, Par. 40.

³ Grove Karl Gilbert: Report on the Geology of the Henry Mountains. U. S. Geog. and Geol. Surv. of the Rocky Mountain Region (1877).

Lake Bonneville. U. S. Geol. Surv. *Mon.* 1 (1890).

⁴ J. W. Powell: Report on the Geology of the Eastern Portion of the Uintah Mountains, and a Region of Country Adjacent Thereto. U. S. Geog. and Geol. Survey of the Terr., 2d Div. (1876).

⁵ Waldemar Lindgren: Igneous Geology of the Cordilleras and Its Problems. Problems of American Geology, Dana Commemorative Lectures (1913).

⁶ F. L. Ransome: The Tertiary Orogeny of the North American Cordillera and Its Problems. Problems of American Geology, Dana Commemorative Lectures.

⁷ Josiah Edward Spurr: Ore Magmas (1923).

⁸ B. S. Butler: Ore Deposits of Utah. U. S. Geol. Surv. *Prof. Paper* 111 (1920).

prove to be of value, the credit is due largely to the inspiration and teaching of the late John Casper Branner. Also due acknowledgment and appreciation is expressed to Bailey Willis and other professors in the Department of Geology and Mining at Stanford University.

An entire volume would be necessary to bring out the details of many of the subjects mentioned; it has, therefore, been necessary to divide the subject matter into two parts. The first is a statement of the history of the geologic events that establish a connection between the broader features of the geology of the Cordilleran region. The second shows the practical application of the knowledge of some of the broader features of the structural geology to certain mining districts of Utah. It involves the correlation of the statistics of production with the structural geology, if such a correlation is possible. This is believed to be possible because not only are many statistical data available, but in few mining regions of the world is the structural geology so well exposed by mountain erosion and by great faults.

MINING DISTRICTS WITHIN THE CORDILLERAN MOUNTAIN SYSTEM OF WESTERN UNITED STATES

Cordilleran Mountain System

The North American Cordillera⁹ extend in a slightly sigmoid curve from Bering Strait to the Isthmus of Tehuantepec, a distance of 4500 miles, with an average width of 500 and 600 miles, extending from the Pacific Ocean to the Great Plains; in fact, this great system forms but a part of the "circle of fire" which encompasses the Pacific.¹⁰ In this paper, however, only that part of the Cordilleran System within the United States, as shown on Fig. 1, will be considered.

General Distribution of Ore Deposits of Western United States

The area through which the western mining districts are distributed is shown on Fig. 1. The distribution and more important features of the geology are given by Lindgren¹¹ as follows:

The most characteristic occurrence of important metal deposits is where intrusive masses of moderate size have broken through the sedimentary series; as a rule, the sediments are of Paleozoic age and the intrusive rocks are granites, diorites, and monzonites of Cretaceous or Tertiary age. In all such deposits, the metallization appears to have taken place shortly after the intrusion. The deposits may develop either in the sedimentary rock or in the igneous rock, and they have been laid bare by deep erosion of the rocks overlying them.

⁹ F. L. Ransome: *Op. cit.*, 287.

¹⁰ Waldemar Lindgren: *Op. cit.*, 235.

¹¹ Waldemar Lindgren and James M. Hill: *The Mining Districts of the Western United States*. U. S. Geol. Bull. 507 (1912), 8.



FIG. 1.—RELIEF MAP SHOWING SURFACE FEATURES OF WESTERN PART OF UNITED STATES.^a

^a Original map is taken from U. S. Geol. Surv. Bull. 611; faults are taken from other-geological literature.

In another less common kind of deposit, the ores are contained in lavas of Tertiary age and may be of great richness. The igneous rocks in this case are usually flows of andesite or rhyolite. These deposits occur in regions like Nevada, where there have been eruptions from many local vents and where the rocks have been extensively decomposed by solution and gases of volcanic origin. Many of these deposits have suffered little erosion since their formation, which followed closely the end of the epoch of eruptions.

Few mining districts are found in the vast plateau province of southern and eastern Utah, western Colorado, northern Arizona, and western New Mexico, already referred to; this barren area also extends into western and central Wyoming. In this great province, the sediments, ranging in age from Cambrian to Eocene, lie in horizontal position almost unbroken by mountain-building disturbances. An analogous condition exists in the enormous areas in the Northwest, in Oregon and Washington, with extensions into southern Idaho, Nevada and California. Here flows of igneous rocks, mainly basalts of Tertiary to early Quaternary age, together with tuffs and lake beds, cover the country, and few traces of mineralization are found in them.

As a rule the deposits occur in mountain ranges, but here again it is necessary to draw a distinction, for many of the most conspicuous and highest ranges are almost void of metallic ores. Among well-known instances of this barrenness are the high Sierras of California, the Cascade Mountains of southwestern Washington, the Front Range of Colorado, the Bitterroot Range of Montana and Idaho, and finally the Wind River and Teton ranges of Wyoming. It is clear, then, that mere uplift, faulting, and crushing, and attendant circulation of atmospheric water are not sufficient to produce metal deposits. The ranges just enumerated consist either of enormous areas of mountainous intrusive rocks of post-Jurassic age, or of Tertiary lava flows and volcanoes, or of pre-Cambrian granite and gneiss.

The distribution of the ore deposits, as described, are closely related to the igneous rock from which they are derived through processes of differentiation accompanying the cooling of deep-seated magmas. The igneous rocks are undoubtedly the result of mountain-building forces that have affected large areas in the Cordilleras during certain periods of deformation.

AGE RELATIONS OF PERIODS OF DEFORMATION AND IGNEOUS ACTIVITY

Little is definitely known of the age relations of periods of deformation in the Cordilleran region up to the end of the Cretaceous, but this much is quite certain: That near the end of the Carboniferous, mountain-building forces were set in operation that have since raised the western part of the United States to its present elevation and outline. That these forces were intermittent is indicated by elevation and submergence of great areas. With each succeeding period of elevation, the land area became larger and the ocean retreated until the present continent was formed. In view of the deformation accompanying the more recent periods of elevation, it is reasonable to believe that it accompanied the earlier also. That the earlier periods of elevation and deformation were accompanied by igneous activity is indicated by the age and distribution of volcanic rocks, summarized from Lindgren's work, as follows:

Periods of Igneous Activity

Igneous activity began in the Cordilleran region far back in the pre-Cambrian. Of this period and the activity in the Cambrian, very little is known; but it is believed that from the Cambrian to the Carboniferous there was a period of comparatively little volcanic activity, in other words, a period of quiescence.

In the Carboniferous, volcanic activity began in a comparatively narrow strip bordering the Pacific, from Alaska to central California. During the Triassic and the Jurassic, this belt was extended from Alaska to Lower California and from the Pacific to eastern Nevada. At the end of the Jurassic, or early in the Cretaceous, came the greatest period of Cordilleran volcanism. At this time, a great chain of batholiths, including the Sierra Nevada, was formed from Lower California to the Alaska Peninsula.

Near the end of the Cretaceous or early in the Eocene volcanic activity was widely distributed over the greater part of the Cordilleran region, extending from the Continental Divide to the Pacific. This period was one of intense volcanism during the Eocene and Miocene and greatly diminished in Pliocene, Pleistocene and Recent periods.

POST-JURASSIC DEFORMATION AND IGNEOUS ACTIVITY

During the greater part of the Cretaceous, there was a long period of quiescence during which neither intense deformation nor volcanism was active. It was during this period, however, that the forces were stored which were destined to produce the Laramide revolution.¹²

The Laramide Revolution

Near the end of the Cretaceous came a comparatively short period of intense deformation throughout the greater part of the Cordilleran region. It is believed to have been caused by a tremendous thrust from the earth segment below the Pacific. The force was one of intense compression, with its maximum effect along the Pacific and throughout the Great Basin and intermountain region. It diminished somewhat near the Continental Divide. The effect of this great force on the structural geology was regional compression, causing metamorphism, folding of the formations and great overthrust faults. The folding and overthrust faulting are only the comparatively shallow surface manifestations of the intense deep-seated compression.

The compression is believed to have been caused by deep-seated rock flowage from beneath the Pacific toward the land area, the rock flowage being caused by subsidence of the marginal area of the more dense segment below the Pacific, the result of loading with sediments

¹² F. L. Ransome: *Op. cit.*, 287.

derived from the land, in accordance with the mechanics of isostasy.¹³

It is the force of the underdrag in the transition zone between the zone of fracture¹⁴ and the zone of flowage that produces crumpling or folding and thrusting of the formations above. The effect of this rock flowage, probably at depths between 50 and 100 miles, is intense compression, which produces regional elevation and an increase of an already high temperature. As long as the combined pressure, that due to gravity and that due to rock flowage, is active, it is reasonable to believe that the rock will remain solid; but with the termination of the period of compression, there is a cessation of the horizontal pressure—this may actually develop into a period of relaxation—and the superheated rocks at depth begin to liquefy and expand, forcing their way toward the surface through channels formed by their own force—mechanical and chemical—or through zones of weakness caused by fracturing and faulting in the overlying rocks. Thus is started a period of volcanic activity, which continues until the excess heat has been expended or until the channels to the surface are closed by another period of compression.

It must be clearly understood that the generation of the heat of volcanism as suggested is not one of friction of overthrust faults, etc., but is the direct result of tremendous pressure on the deep-seated rocks.

The earlier ideas that the earth's interior is molten and that rock deformation is caused by a shrinkage of the interior, with a resultant crumpling of the formations forming the outer crust of the earth, has been almost entirely discarded by those most familiar with this subject. Willis states:¹⁵

In the hundred million years of the legible geologic record, there has been no recognizable shortening of the earth's diameter, whereas there have been many episodes of mountain growth, any one of which may have been accomplished in one or two million years. The time ratio is as one hundred to one, or is even more disproportionate.

The internal heat of the earth is derived from the compression of the interior under the weight of the outer portion. It is thus an effect of gravity. The temperature is very high in the interior and is lowest at the surface.

Volcanic activity is believed to follow, rather than accompany, periods of intense deformation due to compression. Thus, periods of volcanism are confined to periods of relaxation between periods of compression. (This concept is similar to, but differs materially from, the one advanced by Spurr.¹⁶) That such a sequence of events has been the case is indicated by the following.

¹³ Thomas C. Chamberlin: *Diastrophism and the Formation Processes*. XIV. Groundwork for the Study of Megadiastrophism. Part I. Summary Statement of the Groundwork Already Laid. *Jnl. of Geol.* (July-August, 1921) 29, 398.

¹⁴ C. R. Van Hise: *A Treatise on Metamorphism*. U. S. Geol. Survey *Mono-graphs* 47, 187.

¹⁵ Bailey Willis: *Geologic Structures* (1923) 130.

¹⁶ Josiah Edward Spurr: *Op. cit.*, 187-252.

Two Periods of Subsidence and Two Periods of Volcanic Activity

Arnold says¹⁷ (omitting the elevation period of late Cretaceous):

The period of the Tertiary uplift of the last world-wide cycle of diastrophism has been marked by two complete subcycles in the Pacific Coast of North America. The first was begun with gradual submergence in early Eocene, was continued by a gradual elevation in the later Eocene when marine conditions gave place to brackish or freshwater conditions, and was completed by the epoch of uplift and erosion in the Oligocene. The second was initiated by submergence in the Miocene, was continued by the gradual elevation in the Pliocene, when, as in the later Eocene, freshwater conditions supplanted marine, and has been practically completed by the Quaternary uplift which marks the present position of the continent.

The periods of marked elevation were the Oligocene, late Pliocene, and Quaternary; the periods of maximum subsidence were the middle Eocene and upper Miocene; the periods of greatest volcanic activity were the middle Eocene and the middle Miocene. It is noteworthy that the periods of maximum volcanic activity were practically coincident with the periods of maximum subsidence in adjacent areas.

Two Complete Cycles of Magmatic Differentiation

Spurr and Ball find that:¹⁸

. . . the Tertiary lavas of the Great Basin are the representatives of two complete cycles of differentiation of a magma of medium composition into acidic and basic lavas, and probably the end of a still earlier cycle is represented.

The age of the two cycles, as I interpret them, is Eocene and Miocene—possibly a minor cycle in late Miocene or Pliocene; see Fig. 2.

Two Periods of Overthrust Faulting

A chain of great overthrust faults extends from southeastern Nevada to the Canadian boundary, as shown in Fig. 1. It appears to be more than a coincidence that these overthrust faults occurred at two distinct periods—late Cretaceous and Oligocene—that were simultaneous with two main periods of elevation along the Pacific and with two periods of mountain-building in the Cordilleras. The position, movement and age of the principal overthrust faults are found in Fig. 1. Description numbers refer to positions on figure.

1. Areas in Utah mapped in detail, see Fig. 4.
2. San Andreas fault, from Fault Map of California,¹⁹ probably a thrust fault with horizontal movement; possibly normal in part.
3. Sierra Nevada fault,^{19, 20} great normal fault along east front of Sierra Nevada; movement, several thousand feet down on east side: time; uncertain, late Cretaceous to present.

¹⁷ Ralph Arnold: *Environment of the Tertiary Faunas of the Pacific Coast of the United States*, in *Outlines of Geologic History*, by Willis and Salisbury, 227–250.

¹⁸ Josiah Edward Spurr: *Op. cit.*, 228.

¹⁹ Bailey Willis: *Fault Map of California*. Seismological Society of America.

²⁰ Waldemar Lindgren: *Tertiary Gravels of the Sierra Nevada of California*. U. S. Geol. Surv. *Prof. Paper* 73, 41.

TENTATIVE INTERREGIONAL CORRELATION SHOWING PRINCIPAL POST-PALAEZOIC											
ERAS	AGES	QUATERNARY	PLIO-CENE	MIOCENE	OLIGOCENE	BOCENE					
CENOZOIC (Recent Life Era of Man)	QUATERNARY	TAKEN LARGELY FROM STANDARD COAST RANGE SECTION OF CALIFORNIA		GEOLOGIC RECORD FROM REFERENCES INDICATED BY NUMBERS <i>13-p 229</i>		GEOLOGIC EVENTS ALONG GREAT VALLEY OF CALIF.		GEOLOGIC RECORD OF EVENTS ALONG GREAT VALLEY OF CALIF.		FORMATIONS AND GEOLOGIC EVENTS JOHN DAY BASIN, OREGON AND WESTERN NEVADA	AGE AND SEQUENCE OF LAVAS IN THE GREAT BASIN REGION <i>7-p 230</i>
		ALLUVIUM				<i>15-p 228</i>				Cascades Pleistocene fossils 7000 feet.	
		SAN PEDRO		Marked Regional Elevation		Important erosion and denudation since gravel and volcanic ash		Erosion interval; tilting of the Rattlesnake formation; erosion of John Day Canyon		Basalt and occasional Rhyolite	
		MERCED		Folding Elevation		San submergence along line sources estimated to be present at elevations of 1000 or 1200 feet		Rattlesnake formation (Tuffs, gravels, ashly soil and rhyolitic lavas).		Andesite	
		SAN DIEGO		Bay of San Francisco largely volcanic and fresh water formations. Blue tuffaceous sandstones.		Volcanoes active along high Sierra of Range between Coast Range and Coast Range		Unconformity; tilting; erosion		Rhyolite and occasional Basalt	
	MIOCENE	SAN PABLO		UNCONFORMITY		MAXIMUM SUBSIDENCE AND GREAT EST VOLCANIC ACTIVITY		MASCALL FORMATION (tuffs, ashes and possible gravels)		Andesite	
		SANTA MARGARITA		UNCONFORMITY				COLUMBIA LAVA—(Basalt and interstratified basaltic tuffs)			
		MONTEREY		UNCONFORMITY				SLIGHT FOLDING			
		VAQUEROS		UNCONFORMITY							
		SAN LORENZO		UNCONFORMITY							
OROGENIC MOVEMENT: REGIONAL COMPRESSION, METAMORPHISM; FOLDING OF FORMATIONS AND MOVEMENT ALONG OLDER FISSURES; FORMATION OF ADDITIONAL THRUST FAULTS SUCH											
MESOZOIC (Middle-Life Era of Reptiles)	TERTIARY	TEJON		MAXIMUM SUBSIDENCE AND GREAT EST VOLCANIC ACTIVITY		CLARNO FORMATION (Shales, tuffs, andesitic and rhyolitic lavas).		Rhyolite and occasional Basalt			
		MINOR OROGENIC MOVEMENT MARTINEZ						Andesite			
								Monzonite and Biotite Andesite			
								Rhyolite			
	CRETACEOUS	CHICO		Gradual transition between Martinez and Chico indicating no orogenic movements.		CHICO FORMATION (Sandstone and Conglomerates)		INTRUSIONS			
				Peridotite eruptions.							
OROGENIC MOVEMENT: REGIONAL COMPRESSION, METAMORPHISM; FOLDING OF FORMATIONS											
UPPER PALAEZOIC	TRIASSIC	HORSETOWN		UNCONFORMITY		KNOXVILLE		Lower Cretaceous west unconformity on Franciscan		Formation of great chain of batholiths exposed by 'J'.	
	CARBONIFEROUS	FRANCISCAN		Sedimentary and VOLCANIC formations		TRIASSIC FORMATIONS are highly metamorphosed folded sediments and VOLCANIC.		Intense volcanic activity. Inyo Co., California, to southern Alaska; very basic.		California, western Nevada and Oregon; andesite and rhyolite flows, submarine and near old shore line.	

Fig.

4. Wasatch fault, great normal fault along west front of Wasatch Mountains; movement, 10,000+ ft. down on west side; time, active from middle Miocene(?) or Pliocene to present.

5. Hurricane; 6, Sevier; 7, Kaibab; 7A, Thousand Lake;²¹ great series of normal faults (reverse in part) of southern Utah; lengths up to 300 miles; movement, down on west side to 7000 ft.; time, active from late Pliocene to present; these are typical Basin Range faults, many similar faults occurred about the same time.

8. Muddy Mountain overthrust fault of southeastern Nevada;²² movement, overthrust block moved from southwest to northeast displacement, very great; time, post-Jurassic (probably late Cretaceous).

9. Mount Nebo (Utah)²³ overthrust fault; geology not studied in detail; probably of late Cretaceous age.

10. Alta, Columbus, Emma overthrust faults, see Fig. 7; movement, overthrust block has moved from west to east at least 6 miles; age, late Cretaceous. (Willard overthrust²⁴; near Odgen, Utah, probably of same age.)

11. Bannock overthrust fault of southeastern Idaho,²⁵ length probably 270 miles; movement, overthrust block has moved from west to east between 12 and 35 miles; age, late Cretaceous, between Adaville and Evanston of Veatch.

12, 13. Absaroka²⁶ and Darby²⁷ overthrust faults of southwestern Wyoming; movement, overthrust block moves from west to east at least 20,000 ft. on each fault; Absaroka early Eocene, Darby late Cretaceous.

14. Heart Mountain overthrust of Wyoming,²⁸ probably traceable for 150 miles; movement, overthrust block has moved about 28 miles from west to east; late Eocene or early Oligocene.

15. Beartooth overthrust of Montana,²⁹ believed to be continuation of Heart Mountain overthrust; probably same movement and age.

²¹ C. E. Dutton: *Op. cit.*, 25-35.

²² Chester R. Longwell: The Muddy Mountain Overthrust in Southeastern Nevada. *Jnl. of Geol.* (Jan.-Feb., 1922) 30, 63.

²³ G. F. Loughlin: Reconnaissance in the Southern Wasatch Mountains, Utah. *Jnl. of Geol.* (1913) 21, 452.

²⁴ Eliot Blackwelder: New Light on the Geology of the Wasatch Mountains. *Bull. Geol. Soc. of Am.* (1911) 21, 517-542.

²⁵ R. W. Richards and G. R. Mansfield: Geology of the Phosphate Deposits Northeast of Georgetown, Idaho. U. S. Geol. Surv. *Bull.* 577, 37.

²⁶ A. C. Veatch: Geography and Geology of a Portion of Southwestern Wyoming. U. S. Geol. Surv. *Prof. Paper* 56, 109.

²⁷ A. R. Schultz: Geology and Geography of a Portion of Lincoln County, Wyoming. U. S. Geol. Surv. *Bull.* 543, 84.

²⁸ D. F. Hewett: The Heart Mountain Overthrust, Wyoming. *Jnl. of Geol.* (1920) 27.

²⁹ Arthur Bevan: Summary of Geology of Beartooth Mountains, Mont. *Jnl. of Geol.* (1923) 31.

16. Overthrust faults of Phillipsburg Quadrangle³⁰ movement, overthrust block moves eastward; displacement, probably great; late Cretaceous with probably later thrusting.

17. Lewis overthrust of Montana;³¹ mid-Tertiary completed before Miocene; Mansfield places Lewis overthrust in late Eocene or early Oligocene.³²

18. Chief Mountain overthrust fault³³ Montana; movement, overthrust block has moved 15 miles eastward.

19. Lombard overthrust fault and related geologic features.³⁴

20. Osburn fault Coeur d'Alene district, Idaho³⁵ strike a little south of due east; length 300 miles, dip south 60°-70° south; movement, 10 to 15 miles east on north or foot-wall side; late Cretaceous; part of later movements contemporaneous with mineral deposition.

21. John Day Basin.³⁶

22. Overthrust fault of Manhattan, Nev.;³⁷ controlling factor in localizing ore deposition prior to late Cretaceous or early Eocene mineralization.

Oligocene Deformation

The forces producing the Oligocene deformation were evidently transmitted through the area bordering the coast of California without producing marked folding, and it is possible that the force was principally active in the area north of California and generally in the northern Cordilleran region, as indicated by Oligocene folding in eastern Washington,³⁸ slight folding in the John Day formation of Oregon,³⁹ and by the great overthrust faults from northern Wyoming to the Canadian boundary. That a thrust could affect this northern area without materially deforming the southern is indicated by the fact that the east-west Osburn

³⁰ F. C. Calkins and W. H. Emmons: *Geology and Ore Deposits of the Phillipsburg Quadrangle, Mont.* U. S. Geol. Surv. *Prof. Paper* 78, 141-151.

³¹ Bailey Willis: *Stratigraphy and Structure of the Lewis and Livingston Ranges, Mont.* *Bull. Geol. Soc. of Amer.* (1902) 12, 344.

³² G. R. Mansfield: *Structures of the Rocky Mountains in Idaho and Montana.* *Bull. Geol. Soc. of Amer.* (1923) 34, 269.

³³ W. T. Thom, Jr.: *Relation of Deep-seated Faults to the Surface Structural Features of Central Montana.* *Bull. of Amer. Assn. of Pet. Geol.*, 7, 1-13.

³⁴ W. P. Haynes: *The Lombard Overthrust and Related Geologic Features.* *Jnl. of Geol.* (1916) 24, 269-90.

³⁵ Joseph B. Umpleby: *The Osburn Fault, Idaho.* *Jnl. of Geol.* (1924) 32, 601-614.
O. H. Hershey: *Origin and Distribution of Ore in the Coeur d'Alene* (private publication). *Min. & Sci. Pr.* (1916).

³⁶ J. C. Merriam and W. J. Sinclair: *Tertiary Faunas of the John Day Region.* *Bull. Dept. Geology Univ. of Calif.* (1907) 173.

³⁷ Henry G. Ferguson: *The Limestone Ores of Manhattan, Nevada.* *Econ. Geol.* (1921) 16, 1-36.

³⁸ Ralph Arnold: *Op. cit.*, 236.

³⁹ J. C. Merriam and W. J. Sinclair: *Op. cit.*

fault has had an almost horizontal movement of 10 to 15 miles east on the north side. In California, there is no evidence of volcanism in the Oligocene.⁴⁰

Miocene Deformation

Along the Pacific coast, there was a period of widespread intense deformation immediately following the Monterey (see Fig. 2) of the middle Miocene period, but sufficient evidence is not available to determine its effect in the remainder of the Cordilleran region. The Miocene is, however, a period of widespread volcanic activity in the Cordilleran region.

STRUCTURAL GEOLOGIC FEATURES DEVELOPED BY DEFORMATION AND IGNEOUS INTRUSIONS

With deformation and igneous activity due to and the result of mountain-building forces affecting great areas of the Cordilleras, similar geologic structure should be found in many widely separated mining districts; this appears to be the case.

North-South Folding

Approximately north-south trending anticlinal and synclinal folds have been prominently developed, especially in the pre-Tertiary sedimentary formations and lava flows. The folding varies between broad open folds, close folds, and overturned folds; finally, overthrust faults may be formed. Such folds are common to large areas; they are caused by compression and are not often caused by the intrusion of igneous rocks.

In the western Cordilleras, the folding may be the result of early, as well as late, periods of deformation, as already mentioned, and the age may be difficult to determine. In the southern part of the eastern Cordilleras, the folding is due almost entirely to late Cretaceous deformation; while in the northern section, from northern Wyoming to the Canadian boundary, the folding may be due to late Cretaceous, Oligocene, and possibly middle Miocene deformation.

Domical Uplifts

Domical uplifts, including circular, elliptical, or elongated uplifts, are as widely distributed as the intrusive igneous rocks with which they are associated, and are of the same age. Where these uplifts are caused by more or less vertical elevation by the intrusion of igneous rocks from below they are of great importance because of the occurrence of ore deposits in the overlying arched formations.

⁴⁰ Ralph Arnold: *Op. cit.*

The intrusion of the igneous rocks has usually been along zones of weakness that have been formed in the outer portion of the earth during periods of deformation. Along these lines of weakness, the igneous rocks have found their way toward the surface. Often they produce domical uplifts during the earlier part of the intrusion, but eventually the igneous rock may find its way to the surface through fissured zones in the arched sediments covering the larger masses of the deep-seated igneous rocks. If the vent through which the volcanic material comes to the surface is more or less circular in shape, a more or less circular (often flat-topped) dome-shaped uplift will be produced. Some of the best examples of domical uplifts, the result of lacolithic intrusions of igneous rock, are found in the Henry Mountains of Utah,⁴¹ others are described by Spurr.⁴² If the vent through which the volcanic material comes to the surface is a long fissure or fault, the domical uplift is likely to have an elongated shape. One of the best examples of this particular type of structure is believed to be shown by the elevation of the Uintah Mountains. This structure is a rather flat-topped, unsymmetrical, anticlinal fold about 100 miles long and 30 to 40 miles wide. It has the type structure of a lacolith, though elongated in shape; that is, over the central area the formations are comparatively flat. Near the edge of the structure, along the line of maximum flexure, the formations are abruptly folded and, with steep dips, the older formations pass below the younger flat-lying Tertiary formations.

Complex structures are often produced where north-south folds are superimposed on domical uplifts. In many reports, the north-south folds are attributed to the force of the igneous intrusion, but usually they are not so formed.

After volcanic activity has ceased, the domical uplifts are often deeply buried below flows of lava that cover the country for miles. In this event, the structure may appear only as a volcanic cone or mountain range, similar to those formed where the igneous material finds a ready outlet to the surface and produces little doming.

In general, the presence of the lava flows and the particular stage of their erosion give good evidence as to the amount of erosion that has taken place in the region under consideration. The amount of erosion is often of importance in determining the possible future of a district; this will be discussed later.

Relation of Structure to Extent of Mining Districts

If the intrusion is of the central type and has been accompanied by fissuring and mineralization and the uplift is small, the extent of the mining district is likely to be quite limited. On the other hand, if the

⁴¹ Grove Karl Gilbert: *Op. cit.*

⁴² Josiah Edward Spurr: *Op. cit.*, 187.

uplift is elongated and fissuring and mineralization have been intense, a mining district may be developed for many miles in length.

Three such major elongated uplifts have been described in Utah.⁴³ The most northerly extends almost entirely across the state in an east-west direction, beginning near the Colorado boundary. It is marked by the Uintah Mountain uplift and by a similar uplift in the Wasatch Mountains. In the Uintah Mountains, the igneous rocks so far have not been found in quantity outcropping at the surface; but in the Wasatch and Oquirrh Mountains this uplift is marked by a series of igneous stocks, dikes and irregular masses of granite, granodiorite, quartz diorite and quartz monzonite. It is in a narrow zone bordering these intrusions, near the crest of the uplift, that the mines of the Park City, Big and Little Cottonwood, and Bingham mining districts have been found. Another east-west uplift extends from the Tintic mining district to the Deep Creek mining district near the western boundary of Utah; a third is found in the vicinity of Marysvale, Beaver, Frisco, and Newhouse, Utah.

In general, in the eastern Cordilleran region uplifts caused by intrusion usually extend east-west to northeast, the fissured zones parallel their axes, and the fissures in these zones are usually in the northeast-southwest quadrants. In the Sierra Nevada region, however, some of the principal zones have their greatest extent in a north and south direction, in which respect they are parallel to the great batholith of the Sierra Nevada.

Deformation from Pliocene to Present Basin Range Faulting

At the close of the Miocene, or possibly well into the Pliocene, we can picture the Cordilleras as a region presenting many of the topographic aspects of today but with their average elevations much lower.

In the Pliocene was initiated a period of marked elevation and deformation of a nature entirely different from those of former periods. Whereas the early elevations were largely the result of compression, in the Pliocene the compressional force is apparently due only to the ordinary force of gravity and the elevation appears to be due largely to vertical forces. With the beginning of this elevation, compressional forces gave way to those of tension in the zone of fracture and great normal faults of the gravity, or landslide, type⁴⁴ were developed, especially along the present eastern and western margins of the Great Basin. If these faults are entirely due to regional elevation and represent the collapse of an arch⁴⁵ the zone of fracture must extend much deeper than is generally supposed; which may be the case.

⁴³ B. S. Butler: *Op. cit.*

⁴⁴ Bailey Willis: *Op. cit.*, 44.

⁴⁵ Joseph LeConte: On the Origin of Normal Faults and the Structure of the Basin Ranges. *Am. Jnl. Sci.* [Ser. 3] (1889) 38, 257-263.

The dominant structural features, as developed from the Pliocene to the present, are local warping, block faulting of the gravity type, and tilting of mountain blocks. The tilting to the west of the Sierra Nevada and nearby ranges and the tilting to the east of the Wasatch, Oquirrh, and other ranges are marked examples of the deformation of this period, which brings us down to the present. Basin Range faults are seldom mineralized; neither do they furnish an outlet to the surface for igneous material, this strongly suggests that these faults die out in the zone of fracture and flowage; section A-B, Fig. 1.

It is noteworthy that the climate was from warm temperature to subtropical during the periods when compressional heat-producing forces and the resultant periods of volcanism were in operation, and that colder climatic conditions were initiated coincident with Basin Range faulting, when tensional forces became active over large areas in the Cordilleran region. This may give some light on the cause of the glacial period.

PRACTICAL CONSIDERATIONS

It is only after a thorough consideration of the age and character of the different kinds of deformation and their relation to igneous rocks and ore deposits that a practical application can be made of the broader features of the structural geology.

Where mineralization is the result of igneous activity of the Eocene and the mining districts are in a region affected by later thrusting, post-mineral faulting may develop to the extent that mining operations become too expensive to be profitable. This condition would be more likely to develop in formations composed of thin-bedded shales and limestones, etc., than in the more competent massive limestones, quartzites, granites, etc., where faulting may not be developed to a serious extent.

If a mining district has been traversed by Basin Range faults, they will have their particular effect.

It is common to hear the statement, "All this district needs is depth." As a matter of fact, erosion may already have removed the only large ore deposits that ever existed in the district; thus the depth of erosion may be of great importance, as suggested by Lindgren⁴⁶ and Butler.⁴⁷ Local conditions are of utmost importance in any district, but a knowledge of the broader features of the geology may be a great help in giving the proper weight to details.

The data of Fig. 2 have been tabulated and presented with the hope that it will stimulate observation throughout the Cordilleran region and

⁴⁶ Waldemar Lindgren and James M. Hill: *Op. cit.*, 8.

⁴⁷ B. S. Butler: Relation of Ore Deposits to Different Types of Intrusion Bodies in Utah. *Econ. Geol.*, 10, 101-122.

that, by discussion, we may learn whether it has a practical application or not.

SUMMARY

In more recent geologic time, in the Cordilleran region, there have been two main periods of mountain-building caused by compression. The age of one period has been definitely established as near the end of the Cretaceous; the other is believed to have been in the Oligocene.

The forces responsible for the periods of compression originated in the Pacific Basin segment and acted as an eastward thrust; thus toward the Pacific the rock deformation has been more intense, gradually decreasing toward the Continental Divide.

During each period great overthrust faults were formed, largely west of the Continental Divide and extending from southeastern Nevada to the Canadian border. Accompanying each period there was folding of the formations, regional metamorphism, generation of heat, elevation and erosion. The heat generated not only affected the climate for long periods of time but was responsible for the igneous activity during the relaxation periods of the Eocene and Miocene.

The periods of greatest igneous activity were relaxation intervals and, in general, periods of subsidence of the land masses. This tends to explain some of the reasons for the conclusions, reached by Spurr, Ball and others, that the lavas of the Great Basin region indicate that there have been at least two distinct cycles of magmatic differentiation.

Widespread mineralization closely followed the igneous activity of the Eocene and Miocene, and important ore deposits are associated with structures produced by intrusions of these periods. Ore deposits of Eocene age may be materially affected by faults of the overthrust type. Ore deposits of both Eocene and Miocene age may be displaced by faults of the Basin Range type, which extended from the Pliocene to the present.

OQUIRRH AND WASATCH MOUNTAINS, UTAH

In the preparation of the second part of this paper a careful study has been made of the literature on different mining districts of Utah, but most of the data presented are based not only on the ideas of others but on a personal examination of the geology of the principal mining districts mentioned.

Detailed geologic mapping has been done in certain areas of the Bingham, Big and Little Cottonwood, and Park City mining districts, and most of the mines in these districts have been visited (some of them many times) during the past 14 years.

A great deal of mapping has been done in areas quite distant from the districts mentioned so that the maps and sections would bring out a

connected story on the relation of the ore deposits to the structural geology. The results of this work appear in Fig. 4; the area included is about 2600 sq. mi. In view of this fact, the more important details have been mapped with a reasonable degree of accuracy while the more remote and less important areas have been given less attention; in these areas use has been made of the reconnaissance map of Utah compiled by Butler.⁴⁸

The areas indicated as the Bingham or West Mountain, Little and Big Cottonwood, and Park City mining districts have been reduced from the published U. S. Geological Survey maps of these districts by Boutwell,⁴⁹ Butler and Calkins.⁵⁰ In the vicinity of Mercur, the map compiled by Spurr⁵¹ was used. In the region bordering and between these areas, much detailed mapping has been done by myself in both the Wasatch and the Oquirrh Mountains.

I wish to express my appreciation and thanks to Christian Vrang, R. E. Marsell, R. W. Landwehr, Frank Cameron and J. D. Stott for assisting in the field and office work. Also, thanks is given for courtesies extended by many mining operators in the districts mentioned. For the information necessary to bring the statistical charts up to date, I am indebted to V. C. Heikes, of the U. S. Geological Survey; and for the careful labor of assembling these data and making the many computations necessary, I am indebted to my wife. To Rolf Wooley of the Water Resource Dept. of the U. S. Geological Survey, I express thanks for topographic maps and data.

THE STATISTICAL CHART

This statistical chart has been compiled from the data of the U. S. Geological Survey; all the records are official but in the production record for 1924 it was necessary to include some estimates where the returns were not complete. In the early days of mining some of the records were not kept or are not available, but this would have so little effect on the divisions of the diagrams that the diagrams and figures may be accepted with confidence in their accuracy.

The financial history of the more important metal-mining districts of Utah, for the past 60 years, is shown at a glance. As the total value of the production has reached well beyond \$1,300,000,000, the chart furnishes a most important comparison of the districts mentioned. Each district has made its own record and it is on the basis of this record that the geology of certain districts will be discussed. Bingham, Park City

⁴⁸ B. S. Butler: *Op. cit.*

⁴⁹ J. M. Boutwell: Economic Geology of the Bingham Mining District. U. S. Geol. Surv. *Prof. Paper* 38. Geology and Ore Deposits of the Park City District, Utah. U. S. Geol. Surv. *Prof. Paper* 77.

⁵⁰ B. S. Butler: *Op. cit.*, 100.

⁵¹ J. E. Spurr: Economic Geology of the Mercur Mining District, Utah, with Introduction by S. F. Emmons: U. S. Geol. Surv. 16th Ann. Rept. (1896) Pt. 2, 343-455.

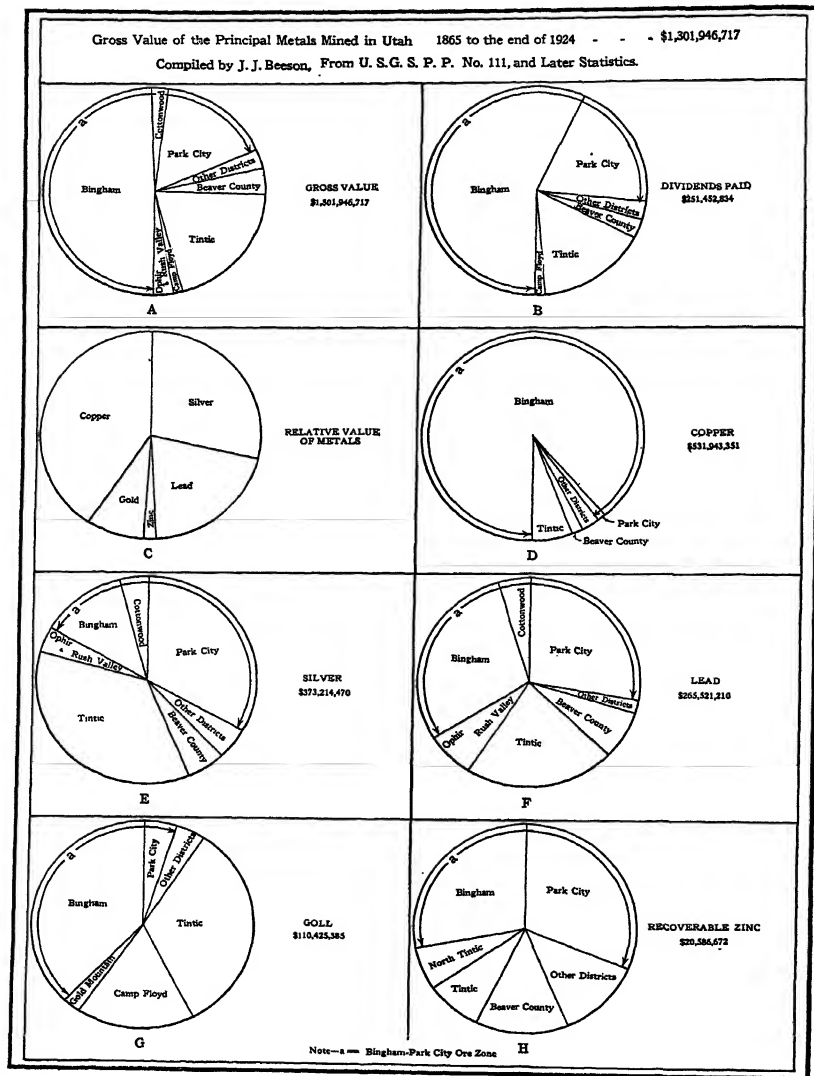


FIG. 3.—STATISTICAL CHART.

and Tintic are the three great mining districts of Utah that have produced the greater part of the production and dividends.

Production from Bingham-Park City Uplift

In order to analyze the production and compare it with the structural geology, the production coming from a single elongated dome-like structure called the Bingham-Park City uplift,⁵² is marked *a*. The mines on this uplift have produced almost 70 per cent. of the gross value of the principal metals mined in Utah from 1865 to 1924 and have paid about 77 per cent. of the total dividends.

Production from the Bingham-Park City Central Ore Zone

At least 90 per cent. of the production and over 95 per cent. of the dividends from the mines along the Bingham-Park City uplift have come from an area 2 miles in width extending through the Bingham, Big and Little Cottonwood, and Park City mining districts. To this area I have given the name "Bingham-Park City central ore zone" and indicated its position on Figs. 4, 5 and 6.

Mines in the Bingham-Park City Central Ore Zone

Beginning at the western end of the Bingham district and extending to the eastern end of the Park City district, a distance of 40 miles, the most important mines located largely within the Bingham-Park City central ore zone are as follows:

Bingham District.—The Utah Delaware, formerly the Utah Consolidated or Highland Boy Mine; Utah Apex; Utah Copper; United States; Ohio Copper; Montana-Bingham, and the Bingham mines; other mines in this zone are in various stages of development.

Little and Big Cottonwood Districts.—The Cardiff, Columbus-Rexall; Wasatch Mines; Alta Merger in part; Emma Silver; Prince of Wales; Michigan-Utah; Woodlawn, and Iowa Copper.

Park City District.—Silver King Coalition, which now owns many properties operated as individual mines in the past; Keystone; Park-Utah Consolidated, which is the result of a consolidation of the Daly Judge, Daly West, Daly, Ontario, Park-Utah, and other properties; also other properties in various stages of development.

Most of the mines mentioned are the result of consolidations of many of the early day properties, and it is this combined production that has contributed largely to the great production and dividend record of Utah.

In considering the area of the Bingham-Park City central ore zone in sections, it is apparent from the statistical chart that the greater part

⁵² B. S. Butler: *Op. cit.*, 100.

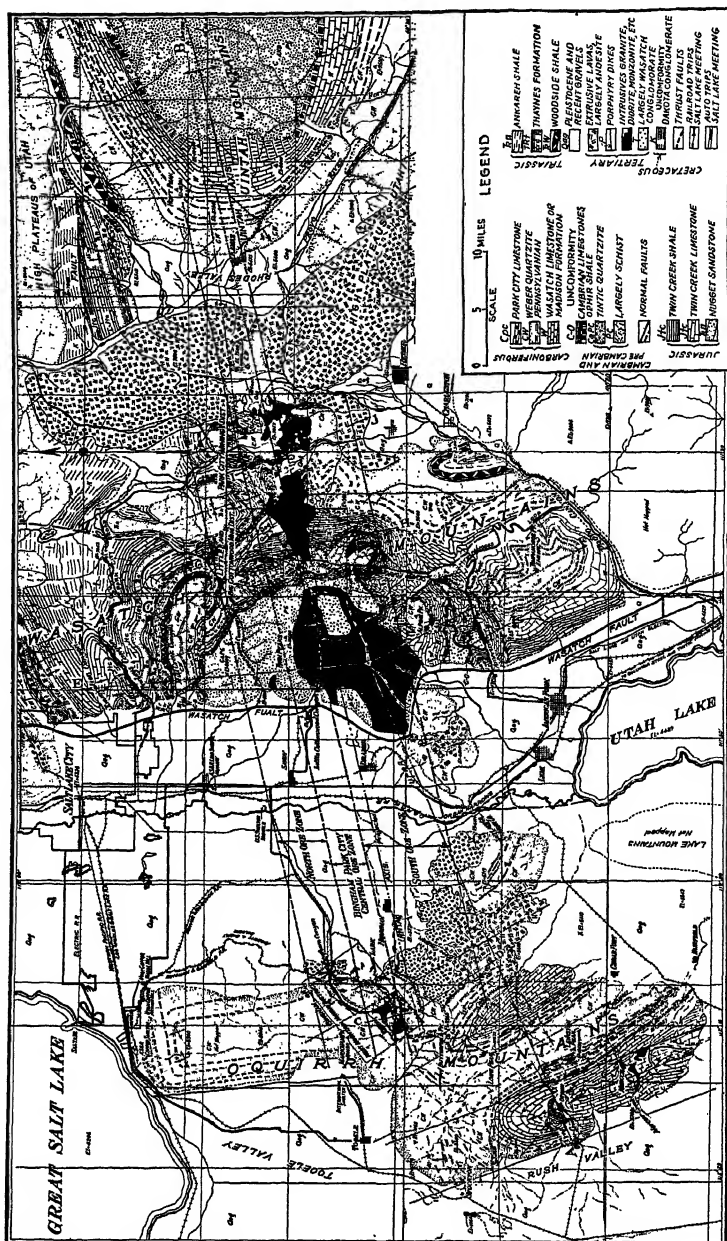
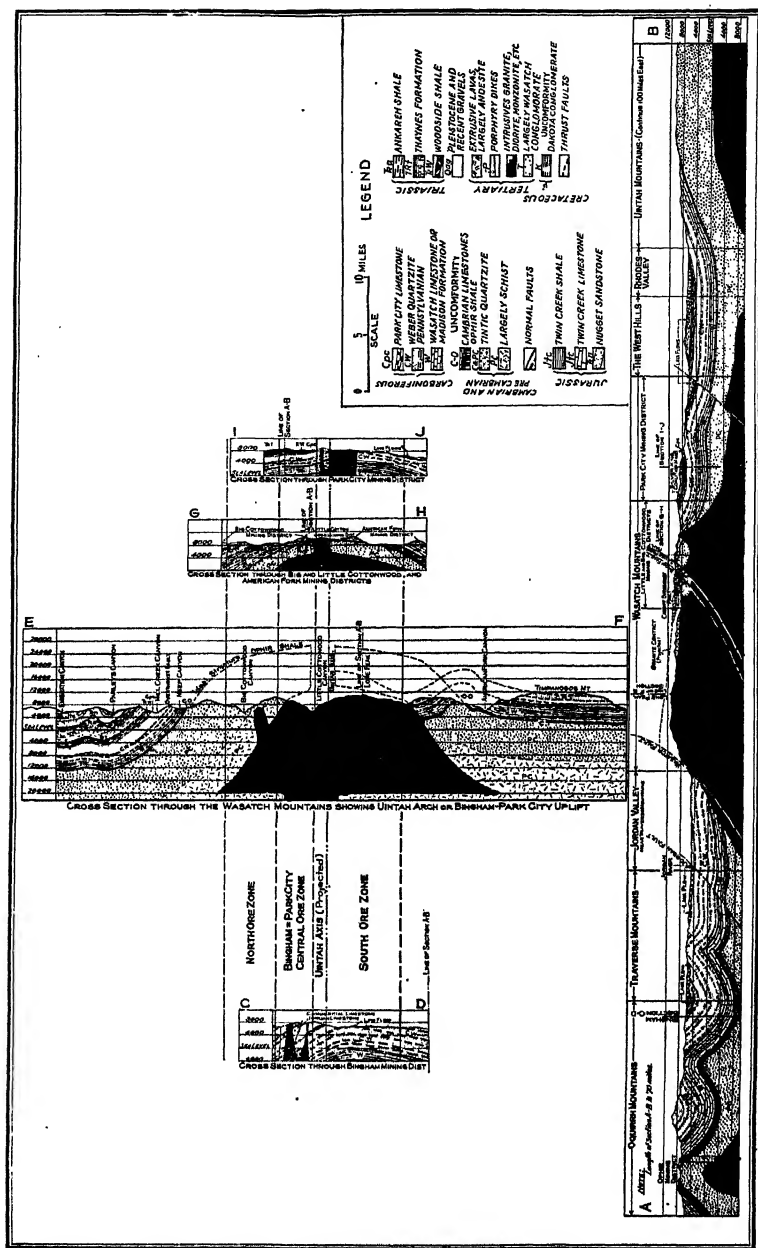


Fig. 4.—RECONNAISSANCE GEOLOGICAL MAP OF PART OF QURRH, WASATCH AND UINTEA MOUNTAINS, SHOWING RELATION OF STRUCTURAL GEOLOGY TO PRINCIPAL MINING DISTRICTS.



of the production has come from the Bingham and Park City districts, while the segment marked "Cottonwood," which includes the record of the Big and Little Cottonwood districts, is very small. This, together with long stretches where there are no mines, immediately raises the question as to whether this is a continuous zone or isolated centers of intense mineralization. That the zone is continuous with a definite relation to the geology will be discussed in the consideration of the structural features of the geology.

STRUCTURAL GEOLOGY

Stratigraphy

From Cambrian to Cretaceous, the sedimentary formations in the area shown on Fig. 4 were laid down without noticeable angular unconformities, though there is a great unconformity or disconformity between the Cambrian and Carboniferous and, in general, the larger part of the Ordovician, Silurian and Devonian is missing.

The details of the sedimentary section in the Oquirrh Mountain are given by Spurr and Emmons,⁵³ Wichman,⁵⁴ Olmstead⁵⁵ and Boutwell,⁵⁶ and in the Wasatch, by Butler and Calkins,⁵⁷ and Boutwell⁵⁸, so they will not be discussed. I have made use of them and have made observations of my own the results of which are shown in Fig. 5, where the correct thickness and position of each formation is shown as nearly as possible. It is noteworthy that the thickness of both Cambrian and Carboniferous increase materially from east to west as shown at A-B. Thus the geologic record from Cambrian is quite largely one of marine sedimentation, with here and there a pronounced change in sedimentation indicating disturbances without marked deformation.

Wasatch Deformation

Near the end of the Cretaceous, between the Adaville and Evanston of Veatch, came the mountain-making disturbances of the Laramide revolution. Forces of compression folded the formations into north-south anticlinal and synclinal folds (see western end of section A-B, Fig. 5) and great overthrust faults were produced (see section A-B central Wasatch

⁵³ Josiah Edward Spurr: *Economic Geology of the Mercur Mining District, Utah.*

⁵⁴ F. M. Wichman: *Ophir Mining District. Eng. & Min. Jnl.* (1920) 110.

⁵⁵ Seymour G. Olmstead: *Economic Geology of the Ophir Mining District. Econ. Geol.* (1921) 16, 433.

⁵⁶ J. M. Boutwell: *Economic Geology of the Bingham Mining District, Utah. U. S. Geol. Surv. Prof. Paper* 38.

⁵⁷ B. S. Butler: *Op. cit.*

⁵⁸ J. M. Boutwell: *Geology and Ore Deposits of the Park City District, Utah. U. S. Geol. Surv. Prof. Paper* 77.

and Fig. 7). This was a period of widespread deformation throughout the Cordilleran Mountain system: it gave birth to many mountain ranges, among which were the original Wasatch and Oquirrh. In view of the widespread nature of this period of deformation, the folding and overthrust faulting of the Wasatch and Oquirrh are correlated with the interval between the Adaville and Evanston, of Veatch, in southwestern Wyoming, with which is also correlated the Bannock and other great overthrust faults. This age of the Wasatch deformation is further indicated by the fact that the north-south folds and overthrust faults are involved in the elevation of the Bingham-Park City uplift, which is the westward extension of the Uintah uplift.

The Uintah Uplift

One of the best descriptions of the great unsymmetrical, flat-topped anticlinal fold that forms the Uintah mountain uplift is by White:^{58a}

The Uintah fold proper is about 150 miles in length, and from 30 to 40 miles in width at the extreme limit of the upturned strata at either side. Its western end is blended with the Wasatch Range in Utah, which it meets nearly at right angles. Its eastern terminus is about 30 miles within and east of the western boundary of Colorado, and about the same distance from the northern boundary. Its axis is not quite straight, the maps of its surveys showing gentle and somewhat irregular meanderings.

Powell has shown that the Uintah fold is of a peculiar type,⁵⁹ being characterized by an upward flexure of the strata at either side, which in some places is a true fault, while between the two abrupt side flexures the fold is broad, and its convexity comparatively gentle.

Such a study of the region round about the Uintah Mountains shows that the formations there were not all equally involved in the great fold. The present condition of those formations and their relation to one another, show that all of them, from the Uintah sandstone (Cambrian) to the Laramie inclusive, were equally involved in that fold, while the Tertiary formations to those older ones, show that these were only partially involved in it.

Time of Fold

From the foregoing, and other correlated facts, the inference is drawn that the elevation of the great Uintah fold was begun immediately upon the close of the Laramie period, and before the first of the fresh-water Tertiary strata were deposited; and that it was nearly complete before the deposition of the Brown's Park group; the latest of the fresh-water Tertiary series.

Thus with the age of the Wasatch deformation definitely established, as between the Lower and Upper Laramie and the overthrust faults of this period involved in the Bingham-Park City uplift, the evidence appears conclusive that this uplift could not have been simultaneous with the Wasatch deformation and must have followed it as did the Uintah uplift, of which it is a part.

The evidence is quite clear that this uplift was developed throughout the greater part of the Eocene, which is coincident with deformation

^{58a} C. A. White: Report on the Geology of the Eastern Portion of the Uintah Mts. U. S. Geog. and Geol. Surv. of the Terr., 1876.

⁵⁹ J. W. Powell: *Op. cit.*

caused by igneous intrusions, which were widespread over the Cordilleran region during this period. This suggests a volcanic origin of the Uintah uplift, which is further strengthened by its peculiar structure and the line of intrusive masses along its axis in the Wasatch and Oquirrh Mountains.

It is reasonable to believe that great quantities of igneous material were gradually added to the laccolithic or batholithic masses below the Uintah-Park City-Bingham uplift during the larger part of the Eocene time. The Uintah uplift continued to grow through the deposition period of the Green River formation, which is involved in the uplift, but had practically ceased before the deposition of the Bridger formation, which shows little effect of this elevation.

The Bingham-Park City uplift probably ceased to grow materially after the middle Eocene, because it is believed that during the deposition of the middle Wasatch formation large areas along the Bingham-Park City uplift and adjacent regions were flooded with lavas from the intrusive masses along the axis of this uplift in the Wasatch and Oquirrh ranges. Thus the effect in this region during later Eocene was probably the formation of a range of volcanic mountains. Remnants of these lava flows still cover large areas east of the Park City and Bingham mining districts, where they rest on the middle Wasatch conglomerate or erosion surface.

If the Uintah uplift had ceased to grow by the beginning of the Bridger time, the volcanic activity of Bingham and Park City had probably begun to subside as in adjacent regions and the magmas had begun to cool in the late Eocene.

Fissuring and Mineralization

No sooner had the magmas of the Eocene, below the Bingham-Park City uplift, begun to cool than the mountain-making forces of the Oligocene were set in operation. In this region there was little, if any, apparent deformation such as folding or overthrust faulting at this time, but it is believed that the forces were sufficient to cause intense fissuring in the solidified portion of the cooling magmas and in the formations elevated above them. This was the most important event in the history of this region for at this time the great fissured zone extending from Bingham to Park City was formed.

Formation of the Bingham-Park City Central Ore Zone

The formation of this great zone of fissuring and mineralization is believed to have been caused by the forces of mountain-making disturbances rather than the contraction stresses of the cooling magma, because the fissured zone is more or less restricted to an area about 2 miles wide and at least 40 miles long and parallel to the Uintah axis. The zone

extends between and through the Bingham and Park City mining districts, as shown in Fig. 4. Its course is about North 78° East. The principal fissures are not parallel to the zone but apparently form a system in the northeast quadrant with the major fissures striking about North 65° East and the minor fissures striking North 35° East.

The reason offered for the localization of this zone of fissuring and intense mineralization is that it occupies a position above the fissure through which the igneous material was added to the laccolith. Thus the mineralizing solutions responsible for the great mines came not only from the cooling of the laccolith but also from the dike extending to immense depths below it. Thus the ore deposits along this central ore zone would be caused by long-continued mineralization. This is suggested to some extent by telescoped orebodies in the Little Cottonwood mining district. The dike occupying the fissure below the laccolith would, especially before solidification, be a zone of weakness along which strains would readily be adjusted through fissuring. This is further indicated by the degree of igneous metamorphism of the lower sedimentary formations in the Big and Little Cottonwood mining districts along the central zone. Also this zone is near the axis or the top of the great uplift. It occupies the area between the Uintah axis and the northern line of maximum flexure, which is also the zone of disturbance in the Uintah Mountains.

Fissuring and Mineralization in the North and South Zones

The North and South Zones are shown in Figs. 4, 5 and 6. In both zones, important fissures are few in number and the mineralization has not been intense except in a few mines from which substantial orebodies have been mined—of course, this does not preclude the possibility of finding other deposits. In these zones the ore deposits have a tendency to form as replacement deposits in the first favorable limestones above the Tintic quartzite; while in the central zone, formations in higher horizons often contain important deposits.

After considering the foregoing data and the information derived from the examination of the broader features of the geology and the details of many mines in different parts of the Bingham-Park City uplift, I believe that the evidence indicates that the outline of the deep-seated igneous mass is quite regular; and, during its intrusion, the sedimentary formations were elevated, forming a great comparatively regular anticlinal uplift; and along its crest intense fissuring and mineralization have taken place. To be sure, the mineralization may have been more intense in certain sections than in others, but the occurrence of many important ore deposits in the Bingham and Park City mining districts is believed to be the result largely of intense mineralization along the

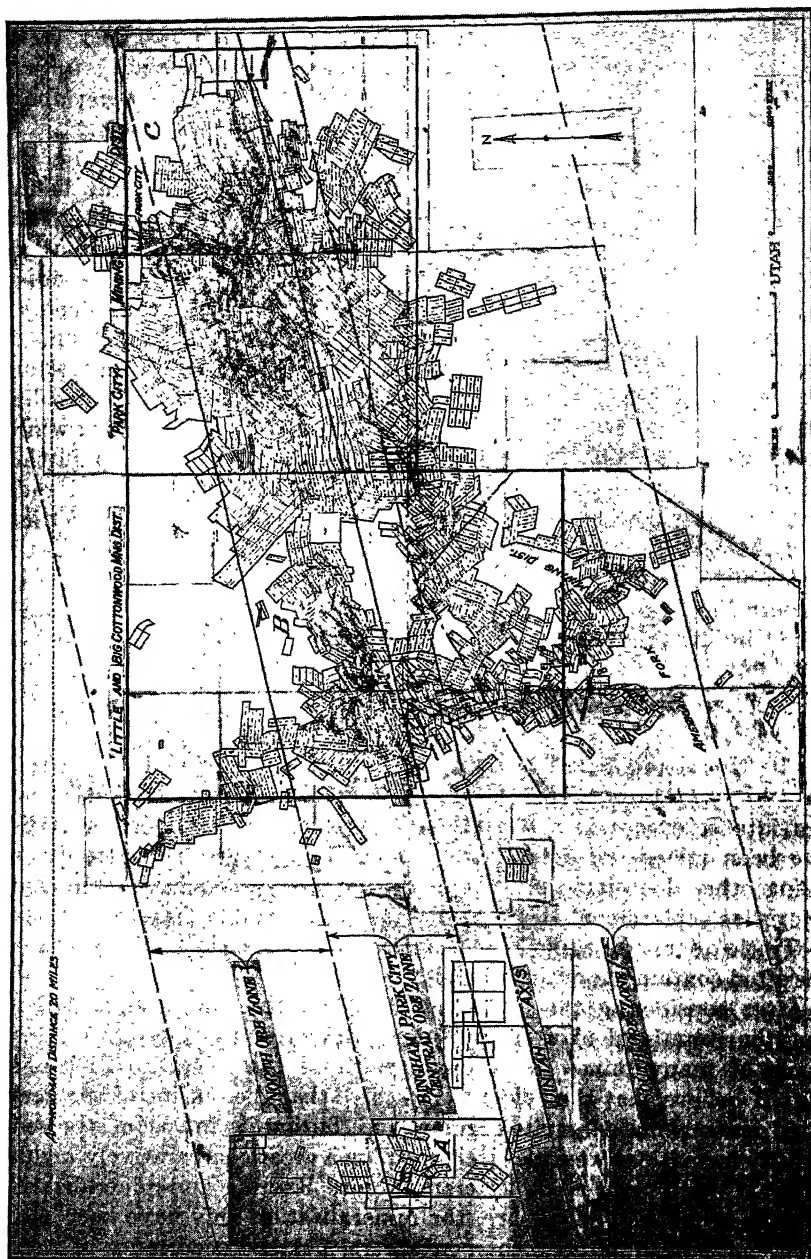


Fig. 6.—APPROVED PATENT SURVEYS OF BIG COTTONWOOD, LITTLE COTTONWOOD, AMERICAN FORK, UINTAH, BLUE LEDGE AND CANYON COUNTRY TERRITORIES COMPILED FROM OFFICIAL PLATS OF U. S. SURVEYOR GENERAL'S OFFICE.

entire central ore zone and their present importance is due largely to the depth of the erosion in these districts.

Middle Miocene Deformation

In view of the fact that the middle Miocene deformation was so intense along the Pacific, there is reason to believe that it may have been responsible for the development of certain structural features in the region under discussion.⁶⁰ Evidence of deformation of this period is believed to be confined almost entirely to faulting and tilting. It is represented by the development of the great Silver Fork and Snow faults in the central Wasatch region, as shown in Figs. 4 and 7 and section A-B, Fig. 5.

The largest of these faults is the Silver Fork fault. It has a strike almost due north-south, an average dip of about 45° west, and a down-throw of about 5000 ft. to the west. The smaller is the Snow fault with a displacement of 1000 ft. The latter develops as a split from the Silver Fork fault about 1 mile north of Little Cottonwood Canyon near Alta. From this intersection south, at least to Alta, they are almost parallel in strike and dip and the formations between them are intensely faulted, as shown in Fig. 7.

The age of the Silver Fork and Snow faults is based on the indications that they are entirely post-mineral, which would make their age later than the mineralization of early Miocene; they must have been formed long before the Pliocene, because there is little or no topographic expression of the existence of the Silver Fork fault where it passes under the Pliocene plateau south of American Fork Canyon.

The Snow fault is peculiar in that the bending of the formations, section A/B, Fig 7, indicate that it first developed as a reverse fault, the hanging wall being thrust upward and eastward, accompanied by an increase in the dip of the formations on the hanging-wall side and probably by considerable tilting and elevation in the section of the Wasatch between Alta and the Wasatch fault. Later, this fault appears to have had a downward movement on the west, or hanging-wall side, until it now represents a normal fault with a displacement of about 1000 ft. The first movement on some of the other faults of the Basin Range type in this region may have been similar.

Evidence of Tilting

Evidence of eastward tilting of the mountain block between the Silver Fork and Wasatch faults is shown in the Little Cottonwood Canyon, not only by the increase in the dip of the formations but by the fact that the overthrust faults of late Cretaceous age now dip about 30° east,

⁶⁰ Hyrum Schneider: A Discussion of Certain Geologic Features of the Wasatch Mountains. *Jnl. of Geol.* (1925) 33, 41-44.

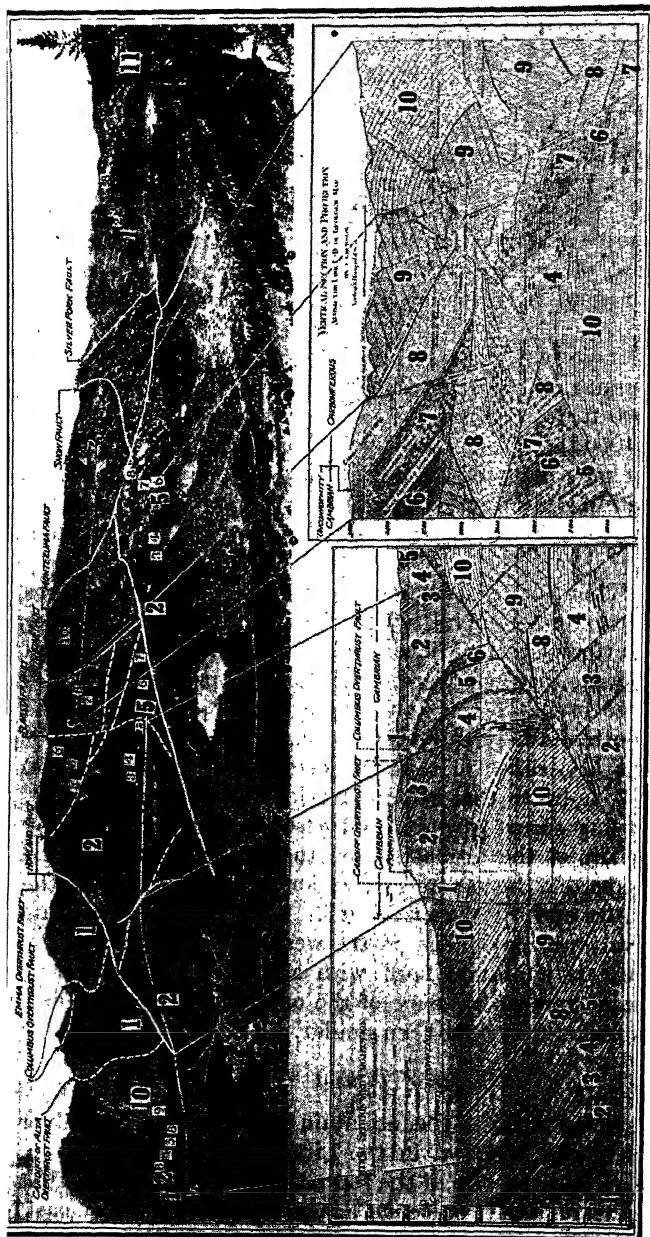


Fig. 7.—PANORAMA VIEW LOOKING NORTHWEST, NORTH, AND NORTHEAST, SHOWING LARGER PART OF LITTLE COTTONWOOD MINING DISTRICT, CENTRAL WASHATCH REGION, ALTA, UTAH. SHOWS FOUR OVERTHURST FAULTS; THE ALTA OR CARDIFF, THE COLUMBUS, AND THE EMMA WHICH SPLITS INTO AN UPPER AND LOWER PLANE; THEIR AGE IS LATE CRETACEOUS (PRE-MINERAL); OVERTHURST FAULTS ARE CUT BY INTRUSIVES OF EOCENE AGE; FISSURES OF LATE EOCENE AGE CUT BOTH IGNEOUS ROCKS AND OVERTHURST FAULTS, FROM WEST TO EAST, ESTIMATED AT 6 MILES; SNOW FAULT DEVELOPED FIRST AS A REVERSE FAULT WITH DOWNTHROW TO WEST OF ABOUT 5000 FT. SNOW AND SILVER FORK FAULTS ARE TYPICAL NORTH-SOUTH BASIN RANGE FAULTS, BELIEVED TO BE ENTIRELY POST-MINERAL, AS ARE ALL FAULTS EXCEPT OVERTHURST TYPE; FORMATIONS 8, 5 AND 3 CONTAIN PRINCIPAL SILVER-LEAD REPLACEMENT OREBODIES; FORMATION 10 CONTAINS LARGE SILVER-LEAD REPLACEMENT OREBODIES IN OVERTHURST FAULT ZONES ONLY; SECTION A-B IS ON SAME LINE AS SECTION A-B, FIG. 5.

whereas they must have had a flat dip to the west during their formation. The contact between the granite and the overlying Cambrian quartzite, west of Alta, appears to dip about 30° to the east and in American Fork Canyon, near Deer Creek, early Wasatch conglomerate has been tilted 45° eastward.

The initial movement on the Wasatch fault may have been of the middle Miocene age, but the topographic features of the present west front of the Wasatch range are the result largely of movements of post-Miocene time.

Evidence of Erosion

The present topographic features of the Wasatch and Oquirrh Mountains, and the mining districts within them, are the results of at least four distinct cycles of erosion.

1. Deformation of the late Cretaceous elevated the Wasatch and Oquirrh Mountains above sea level. During this deformation period, sediments derived from the rapid erosion of these ranges were deposited largely in the Uintah geosyncline, which occupied the area of the present High Plateaus of Utah between the eastern slopes of the Wasatch and the Continental Divide.

2. The Uintah uplift produced further elevation and rapid erosion. By middle Eocene, many topographic features similar to those of today had been developed and lava flows were poured out over a mountainous topography. From middle Eocene to middle Miocene, the Bingham-Park City uplift was protected from deep erosion by lava flows.

3. Further elevation and tilting took place along great faults during middle or late Miocene; and along the western sections of the Wasatch and Oquirrh Mountains erosion removed the lava flows and cut deep into the sediments, removing important ore deposits from the vicinity of the present ore zones. In these areas, by the early Pliocene, the effects of middle Miocene faulting had been largely obscured and a rather mature topography was developed. Evidence of this is seen in the elevated plateau in the vicinity of Timpanogos Peak, as shown in Fig. 8. This plateau has since been warped until it slopes about 3° southerly; its projection over the Bingham-Park City uplift would indicate that erosion up to the time of the formation of this plateau had laid bare a large area of the formerly deep-seated granite now exposed from the central Wasatch to its western front.

4. In the early or middle Pliocene, coincident with Basin Range faulting in southern Utah,⁶¹ and possibly throughout great areas of the Cordilleran region, the principal movement of the Wasatch fault was begun and has continued almost to the present day. The recency of movement

⁶¹ C. E. Dutton: *Op. cit.*, 35.



FIG. 8.—MOUNT TIMPANOGOS OF THE WASATCH. VIEW LOOKING SOUTH SHOWING TIMPANOGOS PEAK AS A MOUNTAIN ELEVATED ON A PLATEAU PROBABLY OF LATE MIOCENE OR EARLY PLOCENE AGE. *A*, ARROWS INDICATE POSITION OF OLD PLATEAU; *B*, APPROXIMATE POSITION OF THE WASATCH FAULT ON WHICH PLATEAU HAS BEEN FAULTED DOWN SEVERAL THOUSAND FEET TO WEST, PRESENT VALLEY IS ABOUT 3000 FT. BELOW PLATEAU; *C*, POSITION OF AMERICAN FORK CANYON, WHICH HAS DISSECTED PLATEAU AS MUCH AS 3000 FT. SINCE THE PLOCENE.

of the Wasatch fault is shown in Fig. 9, where it passes through Pleistocene glacial moraines.

Since the middle Pliocene the Wasatch range has been elevated until many of its mountain peaks rise to between 10,000 and 12,000 ft., while the Oquirrh Mountain peaks are, on an average, about 2000 ft. less.

The deformation on this period has been largely tilting, the result of slight rotation on faults of the gravity type; the Oquirrh has probably been tilted more than the Wasatch.

In this last brief period of geologic time, stretching back to the middle Pliocene, canyons 3,000 ft. deep have been sawed through the west front of the Wasatch. This brings us down to the topographic and geologic structures that are seen along the Bingham-Park City uplift today.

RELATION OF PRODUCTION OF MINING DISTRICTS IN OQUIRRH AND WASATCH TO STRUCTURAL GEOLOGY

The evidence is quite clear that some of the outstanding features of the present topographic and geologic structures of the Oquirrh and Wasatch ranges are the result of block faulting and the tilting of these ranges. The Oquirrh range has been tilted to the east largely by rotation on the Wasatch fault. Thus the elevated portion of this mountain block is the western, which is represented by the crest of the Oquirrh Mountains and the western slopes, while the depressed side of the block lies beneath Salt Lake Valley and along the eastern slopes of the range. Where the maximum elevation has occurred, erosion has cut deep into the sedimentary formations, so that the present surface represents an elevation that was deeply buried at the time of mineralization; in fact, it appears quite probable that the depth was sufficient to produce considerable effect due to rock pressure. Under such conditions fissures would be tight, open cavities would be few, and there would be a less free circulation of mineralizing solutions than in the formations higher up where the pressure was less. This appears to have been the case in the mining districts along the western slope of the Oquirrh range. These districts have not heretofore been discussed, but it is believed that the mineralization was contemporaneous with that in the Bingham-Park City uplift and, in places, this mineralization has been quite intense, as in the Ophir mining district where one property has produced over \$25,000,000 worth of silver-lead-copper ores.⁶² At this particular property, it appears that the mineralization has been very intense and the ore deposits have been formed by the replacement of limestone beds near the top of an anticlinal fold where a concentration of mineraliza-

⁶² Seymour G. Olmstead: *Op. cit.*

tion, together with the presence of the limestones, would make conditions for mineralization almost ideal.

Camp Floyd, Ophir and Rush Valley

In considering the Camp Floyd, or Mercur, mining district; the Ophir and Rush valley,⁶³ or Stockton mining district as representing a mineralized zone, it is quite apparent, as shown in Fig. 3, that the combined production of these districts is very small as compared to that of the Bingham mining district. It is believed that this may be the result of the great depth of these deposits at the time of mineralization.

Bingham Mining District

The Bingham mining district is situated on the eastern slopes of the Oquirrh Range, where erosion has not cut deeply into the sedimentary formations; in fact, it appears from sections *A-B* and *C-D* and the distribution of the extrusives as shown in Fig. 4, that the lava flows have scarcely been eroded from this mining district. Thus, at the time of mineralization, it is reasonable to believe that rock pressure was only moderate; the fissures were more open, there was a free circulation of mineralizing solutions; and, as a result great ore deposits were formed—see descriptions by Hunt⁶⁴ and Boutwell.⁶⁵

The presence of the disseminated porphyry copper deposit of the Utah Copper Co. in the center of the Bingham mining district indicates that mineralization was very intense⁶⁶; but that the presence of the Bingham-Park City central ore zone passing through this district is an important factor in the distribution of the mines is indicated by the fact that all of the long-lived, large-producing mines of the district are within this zone of intense fissuring and mineralization, indicated as the Bingham-Park City central ore zone in Fig. 4. From the Bingham district to the Wasatch Range, the Bingham-Park City uplift is entirely covered by lava flows and valley fill.

Wasatch Front

For several miles east from the Wasatch front there is an area that has been deeply eroded; in fact, so deeply eroded that a large area of the deep-seated granitic mass is exposed, as shown in Fig. 4, and on Section *E-F*, Fig. 5. In this area, the only important ore deposits are the gold

⁶³ B. S. Butler: *Op. cit.*, 362, 366, 382.

⁶⁴ Richard N. Hunt: The Ores in the Limestone at Bingham, Utah. *Trans.* (1924) 70, 856.

⁶⁵ J. M. Boutwell: Economic Geology of the Bingham Mining District, Utah. U. S. Geol. Surv. *Prof. Paper* 38.

⁶⁶ J. J. Beeson: The Disseminated Copper Ores of Bingham Canyon, Utah. *Trans.* (1916) 54, 356.

deposits of the "vein dike" type,⁶⁷ and these have not proved to be of any considerable commercial importance. Nevertheless, they are within the central ore zone, which strongly suggests that the Bingham-Park City central ore zone passes through the deep-seated rocks as well as those arched over the granite. The gold deposits thus mentioned are within area A, Fig. 6. After considering all the evidence, there is good reason to believe that some 25,000 to 30,000 ft. of sediments have been eroded from this area since late Cretaceous and from 10,000 to 15,000 ft. since the time of mineralization; thus these veins are the most deep-seated of all veins in the area discussed.

From the Wasatch front to the Little Cottonwood mining district, there have been few mining properties of any importance, though fissures are found in the granite and in the hard metamorphosed Cambrian quartzites overlying, but seldom is the mineralization strong enough to make deposits of commercial value. Thus we have in the Wasatch Mountains, near the front of the range, a section that has been so deeply eroded that, although the Bingham-Park City central ore zone passes through it, there are practically no paying mines within this area.

Little and Big Cottonwood and American Fork Mining Districts

Where the central ore zone passes through the Little and Big Cottonwood mining districts in the central Wasatch region, erosion has cut moderately deep into the formations, as shown by sections A-B and G-H, Fig. 5. The production record of this district is indicated in the segment marked "Cottonwood" in Fig. 3. In this particular area, the mines are confined largely to the Bingham-Park City central ore zone. Strong, well-mineralized fissures are abundant, but in general the ore deposits are small, compared to the Bingham and Park City mining districts, and there is every reason to believe that the rock pressure, combined with the partial metamorphism of the limestones, has been responsible for the restricted size of the ore deposits. In the mines that have been most productive, the ore deposits are found along exceptionally strong, well-mineralized fissures or along fissures showing less indication of strong mineralization, but where they have encountered exceptionally favorable structural conditions, such as have been produced by the great overthrust faults of late Cretaceous age. Where the fissures intersect these overthrust faults, the mineral-bearing solutions have apparently been dammed by gouge or shear zones and impeded in their progress toward the surface, causing extensive replacement of the limestones by silver-lead ores.

As compared to many mining districts in other sections of the Great Basin, the Cottonwood area has been a reasonably productive region. As compared to the Bingham and Park City mining districts, the pro-

⁶⁷ Josiah Edward Spurr: *Op. cit.*, 707.

duction has been very small; similar conditions exist in the American Fork mining district.

Park City Mining District

Where the Bingham-Park City central ore zone passes through the Park City mining district, there is a zone of intense fissuring and mineralization, within which the great mines of the district have been in continuous operation for the past forty or fifty years. Here again the mining district is situated in the depressed portion of a tilted mountain block where erosion has not been great; in fact, from the presence of the lava flows in the eastern end of the district, occupying a large area farther east, as shown in Fig. 4, and from sections *A-B* and *I-J*, Fig. 5, it appears that erosion has been just sufficient to remove the lava flows from the larger part of the Park City district. Here again, the rock pressure has not been great and there has been a free circulation of the mineralizing solutions; the result has been the formation of great ore deposits.

Fig. 6 is a claim map of the Uintah, Blue Ledge, Snake Creek, Big and Little Cottonwood, and American Fork mining districts. The original of this map is 22 ft. long and 14 ft. wide. It represents the mining claims for which application for patent has been made in the United States Land Office. It is quite apparent that, although the region has been divided into several different mining districts, it is, in reality, one great mining district or mining region. The position of the Bingham-Park City central ore zone, as well as the north and south ore zones, has been shown.

Attention is called to the number of narrow claims that mark the trail of the early miners and prospectors. The abundance of these in the Bingham-Park City central ore zone indicates that mineralization along this zone was apparent on the surface in many places.

Summary and Conclusions

By comparing the production records on the statistical chart, Fig. 3, with the geologic sections, Fig. 5, the practical application of the broader features of the structural geology are clearly indicated. Section *A-B* shows the tilting of the Wasatch and Oquirrh Mountain blocks and indicates the areas of deep and shallow erosion, while cross-sections *C-D*, *E-F*, *G-H* and *I-J* indicate the relative stages of erosion in the region through which they pass. The most important mining districts are found where erosion has been only moderately deep; in fact, just deep enough to remove the lava flows. This condition exists in the Tintic district as well as in the Bingham and Park City districts; these are the three great districts of Utah.

The recognition of zones of fissuring where mineralization is intense and other conditions favorable may lead to the discovery of other great

mines. In the districts mentioned, the important ore deposits belong to a type found at intermediate depths.⁶⁸ In other districts, deposits formed at less or greater depths may be important. In short, each mining district presents problems peculiar unto itself, and these must be given due consideration in applying interpretations of the broader features of the geology.

The future of the mining industry of the West will depend largely on the development of the known mining districts along geologic lines; and in this particular field a consideration of the broader features of the geology will be of utmost importance.

DISCUSSION

C. KEYES, Des Moines, Iowa (written discussion).—The reference, without example, of the origin of the Desert Ranges to segmental rotation, rather than to normal faulting premised by the Gilbertian hypothesis of Basin Range structure, is unfortunate, notwithstanding that it probably marks the passing of the long-assumed explanation of Desert Range genesis. As this phase of the paper has such far-reaching bearing on scientific prospecting for ore deposits, it merits much more careful and fuller analysis and broader consideration, for the Gilbert hypothesis leads mining men to take the wrong trail.

When presenting a new hypothesis of the larger tectonics of the Great Basin, that of the segmental rotation of the mountain blocks more than a mere statement should be given. The statement should be amply supported, in the beginning at least, by abundant concrete example. It should demonstrate wherein all other explanations are unavailing and should preclude the claims of present relief through special or characteristic desert erosion. Theoretical and doctrinaire as all of this may seem, it is important for the future of ore exploration over an area equal to one-third that of the entire United States.

Whether the hypothesis of rotating mountain blocks is the true explanation of Desert Range orogeny remains to be tested in the field. In the presentation of the idea without supporting evidence the theory does not pass beyond the speculative stage and, hence, is on a par with other hypotheses covering the same ground. Failure to produce a concrete example after fifty years of search is the rock upon which the Gilbertian hypothesis foundered.

The only concrete examples of Basin Range structures yet revealed in no uncertain terms prove to be in the nature of overthrust, of which the Sierra de los Caballos, in New Mexico, and the Muddy, Spring Mountain, and other ranges, in Nevada, are typical illustrations. Revelations of the latter, wholly unexpected as they are, may develop into the most important relation of all between structure and ore deposition. Too much stress cannot be placed on the tectonics as an explanation of the present mountain relief of the vast Great Basin region, as this is unquestionably dominantly erosional, of a sort more rapid, more prominently differential, and more potent than any ordinary land depletion and lowering by rain or river, and one especially characteristic of regions with excessively dry climate.

After pointing out the way to genetic association of ore deposition with local tectonics it is necessary to group and correlate the different classes of larger structures so that broad generalization may be indulged in which will be of basic service in the future exploration in the Desert ranges yet untouched. This is, of course, difficult but it is well worth the while in the interests of the mining industry.

⁶⁸ Waldemar Lindgren: *Mineral Deposits* (1913 Ed.), 573.

At present the direct association of the orebodies with the Uintah laccolith is of first importance. Although stratigraphically beneath Cambrian strata, and hence commonly regarded as of pre-Cambrian age, Gickie long ago showed that such intrusions, in other parts of the Wasatch Range, are of relatively recent date. Establishment of the fact that the granitic mass is laccolithic in character is the most promising genetic feature determined in connection with the Utah ore deposits. It should be exploited to the utmost.

A larger aspect, which the author does not bring out sufficiently, is the circumstance that the great Uintah laccolith, unexposed and unknown in the Uintah Mountains themselves, is a result rather than a cause of local doming, a view quite at variance with Gilbert's original idea of such structures. That laccolithic masses are such because of previous sustaining conditions in the strata due to folding or faulting pressures, is premised for the laccoliths of other regions, notably some of those at the southern extremity of the Rocky Mountains in New Mexico, where they also are the immediate source of ore deposits.

C. A. PORTER, Philadelphia, Pa. (written discussion).—The idea of ore zones is not new. It was first called to the writer's attention in 1906, by a Colorado prospector, who said that the ores of Colorado were in zones that extended from the northeast to the southwest. That statement has since been repeated, with modifications, by men with experience in the same country. In Nevada and Arizona, this idea has been expressed more frequently with reference to the zone, or belt, that extends from the Comstock lode into Mexico. The mere possibility of being able to draw straight lines from one district to another does not establish ore zones however. In Nevada, a line may be drawn from the Goldfield to the Austin districts, a distance of about 120 miles passing through Tonopah and Manhattan, making four districts in all. Genetically there appears to be little connection between the districts, other than must be common to the deposits in general in the Basin region; it is even doubtful if they are all in the same north and south line of fracturing, for apparently they lie in two different mountain ranges.

In the Bingham-Park City zone, the deposits of Bingham and Park City lie in a line of weakness or stress, which is recognized by Beeson as a continuation of the Uintah uplift. The Uintah uplift does not cease with the Oquirrh range but extends to the Stansbury mountains, the next range to the west, giving it a length of about 175 miles, the major portion of which is east of the Wasatch, a section of over 100 miles that shows no important ore deposits. In addition, a stretch across the Salt Lake valley is barren and there are no deposits westward from the Oquirrh range, that have yet proved of any great commercial value. There must, then, be some vital difference between the part of the Uintah uplift that is barren and the part that has proved productive.

The answer to this is difficult. The Uintah uplift, from its eastern end, is unbroken by cross ranges until it reaches the north-and-south trending Wasatch. As the latter is entered in the vicinity of Park City, two lines of disturbance meet, and it should require little reflection to realize that at such an intersection there should be a region of intensive fracturing and extreme crustal weakness. Especially, is this true, where both lines of fracturing continue past the region of intersection.

This explanation ignores the presence of igneous rocks. It has been the custom, for many years, to account for the occurrence of ores by the action of igneous agencies. However, if it is pertinent to ask why ore occurs in one section to the exclusion of vast areas about it, it is just as pertinent to ask, why igneous rocks have selected one spot to the exclusion of numerous others; therefore, in attributing ores to the presence of igneous rocks and leaving the presence of the igneous rocks unexplained, we have progressed but little. In fact, one might legitimately reason that this very association

of igneous rocks and ores might be but the outgrowth of a common cause. However, if we assume that igneous or magmatic material is present, what better conditions could be imagined than the locus of two intersecting fracture zones for its escape to the surface? That the entire Uintah fold should display no marked igneous action and should contain no important ore deposits, except where it passes through the Wasatch and the Oquirrh ranges, should attract comment in itself. To ignore the influence of the Wasatch and the Oquirrh faulting, is to close the eyes to plain evidence. We are told (page 771) the "Basin range faults are seldom mineralized; neither do they furnish an outlet to the surface for igneous material, this strongly suggests that these faults die out in the zone of fracture and flowage." That Basin range faults are seldom mineralized is probably true, that is, where they are not complicated with cross faults. His observations on this point has led the writer to expect the same condition for all faults, or lines of parallel faulting, where the adjacent country has not been unstabilized by cross faults.

Where two fracture zones intersect, one of which shows such a large displacement as the Wasatch fault, which is given by Beeson as 10,000 + ft., it becomes difficult to

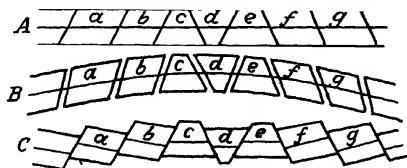
FIG. 10.⁸⁹

FIG. 11.

visualize just what will happen among the resulting fault blocks. Fig. 10, reproduced from LeConte's "Elements of Geology," is pertinent here. LeConte points out that the blocks *d* with the smaller bases would tend to sink while those with the larger bases *c*, *e* would tend to rise. He assumed that a softened and plastic rock underlay the blocks. Today, with our understanding of rock flowage, LeConte's assumption finds a more definite and comprehensible basis for belief. It is not difficult to apprehend that, in a region of intersecting fracture zones, numerous blocks will be formed, which will have unequal bases and unequal loads upon those bases. Also, it seems that these blocks must extend downward until a depth is attained where the compressive strength of the rock will be exceeded. At this point, the blocks exerting the greatest pressure per square area will sink, pushing the flowing rock from beneath their bases. According to hydrostatic law, the increased pressure will be distributed equally through the mass of flowing rock, so that the broad bases, with the lighter pressure per square area, will be forced upward.

It is not difficult to realize that such a region of intersection will be one of extreme instability. Probably no condition can be conceived in which the earth's crust could be more subject to unsettlement and disturbance, nor to offer more favorable conditions for the action of intrusive masses. Indeed, it is conceivable that the conditions may produce intrusions in their entirety; and it is not impossible that this rise and fall of rock blocks may furnish the heat for the magma that fills such a large place in latter day geological speculations.

Suppose that one flank of a mountain range is the result of a line of normal faulting, consisting of, say, four parallel fault planes, as in Fig. 11. Suppose, also, that there has been a total vertical movement of, say, 5000 ft. Obviously, in the vertical movement an open space of 5000 ft. cannot be left beneath the mountain; hence at some depth, rock must have flowed beneath the uplift.

⁸⁹LeConte: Elements of Geology.

Suppose, now, that a second mountain range, which, for simplicity, may be similar to the first, intersects the first at right angles, a complication of results will follow. If, first, we consider the west flank of a north-south range; a sectional view will show the rocks divided into thick sheets by the fault planes and that the tops of these fault planes are not directly above their bottoms, but are to the left, or to the east of the bases of the resulting sheets of rock. The point to be observed is that the sheets are unbalanced and are partly supported by the core of the mountain, or by an intervening sheet. The fault planes of the intersecting range will cut across the sheets of the first range, but the resulting blocks will run diagonally up and down the faces of the second fault planes—the top being east of the bottom; the second set of fault blocks will be unbalanced and the blocks will lean toward their own center—toward the south if the north side of the intersecting range is considered. The final result will be that the blocks will lean toward the southeast and will be in a highly unstable position and subject to unsettlement with every movement in either range.

If we revert to the completion of the first range, assuming that the second has cut the first with its newly formed fault planes but has just begun its growth; we will find the beginning of conditions that are the essence of mineral districts. If the older range has not been settled for a long period, at least part of the heat generated beneath it will be present, especially if its elevation was due to the movement of magma pressing from below.

With the beginning of movement in the cross range, more heat will be generated, or added, if intrusives have originated the second disturbance. Then the settling and shifting of blocks will begin again. As many of these blocks will have unequal bases, compared with their weight, and are unbalanced, such pressure as exists, will not be evenly distributed. Now, we have a succession of movements that must produce heat, and which must be repeated with every uplift of the growing range. There must be an adjustment and a readjustment of the blocks of rock and an intermittent reopening of fissures; an unobtrusive, but an important point in the formation of ores. As the process advances, the flowing rock, where the greatest activity occurs near the bases of the blocks, will increase in temperature and fluidity. This will result in its being pressed quickly into newly opened fissures, and, in many cases, forcing the walls wider apart. In this upward movement of what, sooner or later, must become freely flowing rock, space is vacated. This must immediately be filled by flowage. If the escape of rock is pronounced, the movement below will be in proportion, with probably deep-seated rocks making their way from deeper portions of the flowage zone. If the heating proceeds to the point where lavas reach the flow over the surface, the flowage becomes of great volume and immense quantities of heat are developed, the conditions, then, being such as would be capable of displaying all the phenomena usually attributed to magmatic action.

It is realized that this sketch may appear too definitely drawn and possibly somewhat fanciful. The subject, however, is necessarily speculative, and what has been advanced, has been more of a suggestion to stimulate thought, as the possible circumstances arising out of the general condition are of the greatest variety. It will be noted, that the principles on which the reasoning is based are few and exceedingly simple. The connection between the Oquirrh and Wasatch deposits was first observed by the writer as early as 1910–11. The cross-fracture theory followed and, during the next ten years, numerous other examples were observed in the Basin region. An outline of some of the different types of cross-fracturing studied during that period, among them being those along the Uintah fold, was published in the *Engineering and Mining Journal-Press*, Oct. 25, 1924.

With regard to the development of heat by crustal movements, it is quite evident that for every 778 lb. of the earth's crust that sinks 1 ft., overcoming the internal resistance of rock that is forced to flow, that the heat equivalent for the energy represented

in a fall of 778 ft.-lb., or 1 B. t. u., must be transferred into heat at the point of resistance namely, the flowing rock. When the weight of, say, a mile square of rock, extending to a depth of 30,000 ft., is considered, it is seen that immense quantities of heat are evolved in a movement of 1000 or, perhaps, 5000 ft., and when it is remembered that the specific heat of quartz is about 0.2, that is, 1 B. t. u. is sufficient to raise 5 lb. of quartz 1° F., it is seen that such rock movements may, if necessary, easily account for the entire heat of a very pronounced volcanism.

In the cross-fracture theory, we have a specific cause for the occurrence of mineral districts at a particular place. This necessarily conflicts with any theory of zones. Beeson calls attention to the disturbance extending from Tintic to Dugway, and to a similar occurrence across southern Utah from Marysvale to Newhouse. These, like the Uintah disturbance, give the impression of a more or less continuous chain of mineral districts. It should be observed, in the light of the intersection theory, however, that the Cordilleran system has produced a great number of north-south fractures. These, in themselves, are mostly parallel, and may continue for long stretches without an intersection; but this is not true of the east and west fractures. A fracture of the latter type, 100 miles long, will, almost anywhere in the Basin region, meet with from one to half a dozen north-and-south ranges and is, hence, likely to produce as many mineral districts, or, at least, districts that show some degree of mineralization.

Relations of the Disseminated Copper Ores in Porphyry to Igneous Intrusives

W. H. EMMONS,* MINNEAPOLIS, MINN.

(New York Meeting, February, 1927)

THIS paper is the third of a series treating of the relations of ores of the metals to igneous rocks. The first¹ outlined the general problem; the second² proposed a classification of lode ores, and showed how the distribution of the deposits is in a certain measure controlled by the contour of the roof of the parent batholith. A number of mining districts were discussed, the examples being selected chiefly from group 2 of the classification.

The proposed classification of lode deposits, briefly stated, is as follows:

1. Cryptobatholithic: near hidden batholiths which have not yet been exhumed by erosion.
2. Acrobatholithic: in and near cupolas or domes of batholiths—the high points of larger underlying masses.
3. Epibatholithic: on batholiths, near their rims, but below eroded cupolas, or below the highest points of the roofs.
4. Embatholithic: among closely-spaced outcrops which probably are parts of a single batholith. The outcrops of invaded rocks predominate.
5. Endobatholithic: in and near roof pendants of large batholiths. The invading rocks predominate.
6. Hypobatholithic: in deeply-eroded batholiths, where even roof pendants have been eroded.

Deposits of certain metals on the roofs of the batholiths are much more numerous in some positions than deposits of other metals, and the horizontal zonal arrangements of lodes of different metals are shown characteristically only in certain groups. In a large measure, also, the character of the fracturing and metallization is determined by the position on the roof of the batholith.

The disseminated copper ores in porphyry constitute a clearly-defined type. All the deposits of these ores, as far as known, are in group 2. They appear to have been formed only in and around small upward projections of the parent granitic masses, and not low on the walls of the

* Director, Minnesota Geological Survey.

¹ W. H. Emmons: Primary Downward Changes in Ore Deposits. *Trans.* (1924) 70, 964.

² W. H. Emmons: Relations of Metalliferous Lode Systems to Igneous Intrusives. *Trans.* (1926) 74, 29.

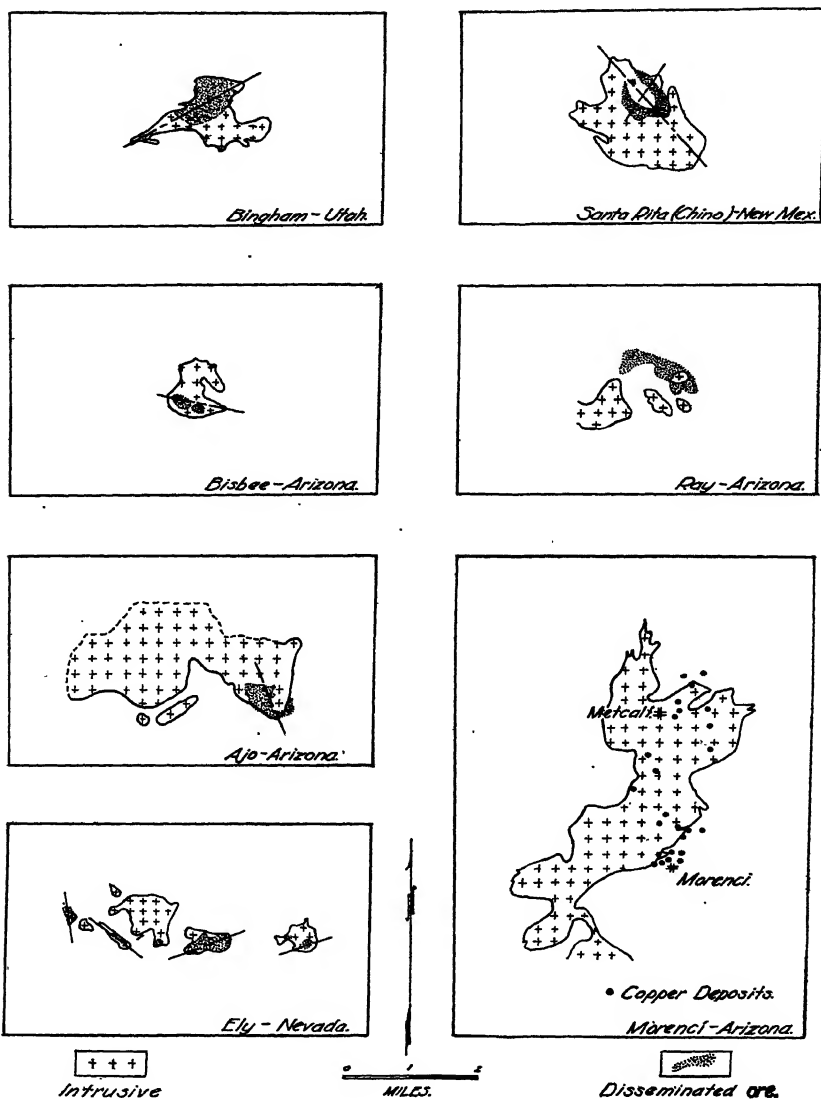


FIG. 1.—MAPS SHOWING DEPOSITS OF DISSEMINATED COPPER ORES AND ASSOCIATED INTRUSIVES. (ALL TO THE SAME SCALE.) BROKEN LINES SHOW AXES OF RIDGES ON CUPOLAS.

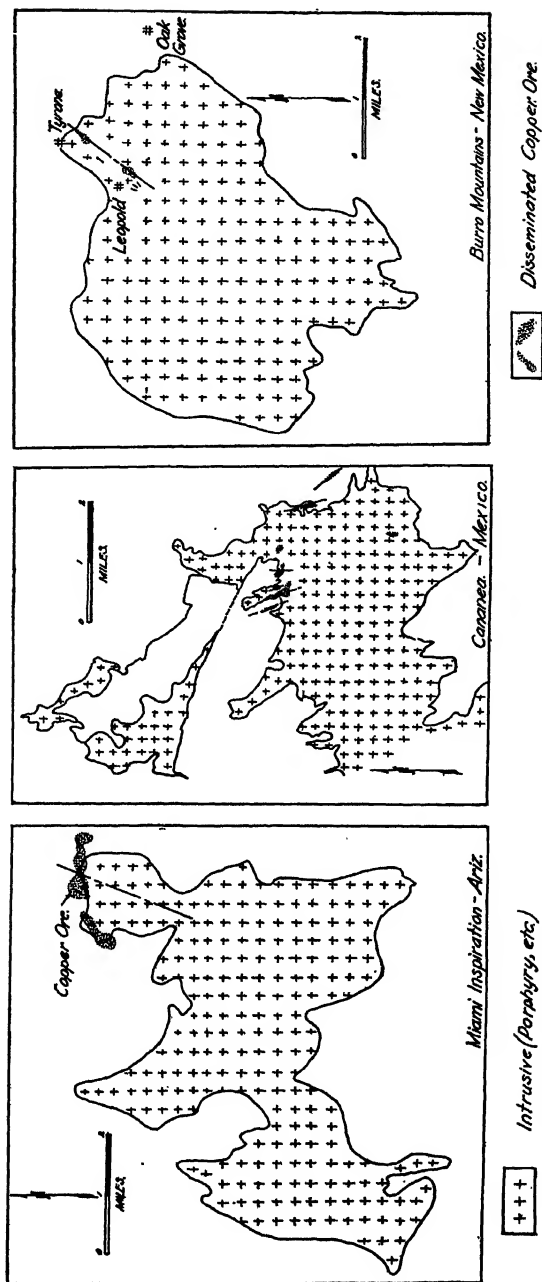


FIG. 2.—MAPS SHOWING DEPOSITS OF DISSEMINATED COPPER ORES AND ASSOCIATED INTRUSIVES OF THREE DISTRICTS. (ALL TO THE SAME SCALE AS FIG. 1.)

intrusive. As a result of this feature of their genesis, the disseminated copper ores of this type occur only in and near small outcrops of the parent masses. This relation is illustrated by Figs. 1 and 2, which show the chief disseminated deposits of copper ore in porphyry for 10 copper-mining districts. The intrusives to which the ores are evidently related are also shown.

HISTORY OF DISSEMINATED COPPER ORES IN PORPHYRY

In general, the history of the deposits of disseminated copper ores in porphyry is as follows:

1. The intrusion of a magma and its solidification, forming diorite, quartz diorite, monzonite, quartz monzonite, or granite, or the fine-grained porphyritic phases of these rocks. The deposits are not known to be genetically related to intrusives as basic as normal diorite.

2. The occurrence of great bodies of fractured or shattered igneous rocks. The fracturing was probably due to stresses, as the most highly fractured areas are confined to certain parts of the igneous masses and are not everywhere apparent. Some of the fractures may be cooling cracks. The shattered and mineralized areas are at the high points of the roofs of the parent batholiths.

3. Primary metallization of porphyry by ascending hot waters emanating from deep-seated sources, probably from a related igneous body not yet completely solidified; formation of protore consisting of pyrite, chalcopyrite, and other metallic minerals, accompanied by the production of sericite from feldspar and other minerals of the porphyry, and by the deposition of quartz. The sulfides fill the minute and closely-spaced cracks, and have been deposited also by replacement as shots and small masses between the cracks. The copper content of the primary ore only rarely exceeds 1 per cent. and is generally not more than $\frac{1}{2}$ per cent. of the total metallized rock or protore.

4. The top of the metallized porphyry body is exposed by erosion to weathering. Waters dissolve the copper from the rock near the surface, carry it downward as copper sulfate, and deposit it near and below the water level in the reducing zone, where the oxygen of the air is excluded. The copper is generally deposited as chalcocite, although some covellite, bornite, and chalcopyrite may be deposited as secondary or supergene minerals. The sulfate waters alter some of the sericite and feldspar to kaolin and other secondary minerals. The chalcocite zone is a thick blanket-like body lying between the leached and oxidized zone above and the primary ore below. It is extensive, undulating, and of irregular thickness and grade. The larger and richer orebodies are in the highly-fractured and shattered parts of the rock, where downward-moving waters converge.

5. The outcrop, the leached and oxidized zone, and the zone of secondary sulfides move gradually downward; as the country is eroded, the copper ore becomes concentrated more and more in the secondary sulfide zone. Pyrite and chalcopyrite are replaced by secondary chalcocite and associated copper sulfides. The primary ore, which originally contained about 0.5 per cent. copper, is converted to an ore with 1.5 per cent. or more. The surface zone is leached, and generally carries only a little copper; iron oxide is commonly present as hematite and limonite; but iron is removed from the deposit, and the leached zone and outcrop gradually become less ferruginous. Silica is concentrated with kaolin at the surface.

MAPS SHOWING DEPOSITS

Geologic maps are issued for nearly all districts containing the disseminated copper ores, and maps of the deposits also are available, but, generally, the deposits are not shown on the geologic maps.³ The maps showing the outcrops of the intrusives are reduced to the same scale in Figs. 1 and 2 and, where practicable, the outlines of the areas of deposits of disseminated ore are superposed on the same maps. All the intrusives to which the ores are related are probably high points in larger underlying intrusives. The centers of the deposits of disseminated ores are not in the centers of any of the larger intrusives, but at the edges of the outcrops, and the Ray and Miami deposits are mainly outside of the invading rocks in the invaded schists. In several of the districts the disseminated ores are near the axes of outward-pointing fingers of the intrusives and in small outlying masses that by further erosion will become fingers.

MINING DISTRICTS

Bingham, Utah.—In the Bingham district, the Bingham stock, which is the rock associated with the deposits, is a monzonite which at places carries considerable quartz and passes into quartz monzonite and monzonite porphyry. At other places it passes into diorite. The maximum surface exposure of the rock is about 2 m. and the greatest width is about 1 m. The disseminated copper deposit occupies nearly a third of the outcrop. The age of the intrusive and of the deposits has not been determined exactly. The intrusive⁴ is Post-Carboniferous and, on account of its general character, is believed to be of the same age as certain other intrusives in Utah which are known to be Tertiary. The ore mined contains less than 1.4 per cent. copper and from 20 to 30 c. a ton in gold and silver.

³ See acknowledgments, page 13.

⁴ J. M. Boutwell: *Economic Geology of the Bingham Mining District*. U. S. Geol. Survey, *Prof. Paper* 38 (1905), 418.

Santa Rita, N. Mex.—The deposits of the Chino mine at Santa Rita⁵ are in a mass of quartz monzonite porphyry. The outcrop of this mass is nearly circular and about 2 m. in diameter. The rock containing the deposit breaks through Paleozoic and Cretaceous limestone, and is probably of late Cretaceous age. The deposit was peneplained before the Tertiary, which places the age of the intrusive and of the ores as late Cretaceous or early Tertiary.

Bisbee, Ariz.—In the Bisbee district, the disseminated copper ores are in the granite porphyry mass of Sacramento Hill, which breaks through Paleozoic limestone and is covered by Cretaceous sedimentary rocks. The exposed area of the intrusive is about 1 m. in diameter, and the area of the disseminated ores is about $\frac{1}{2}$ m. long and $\frac{1}{4}$ m. wide. The deposits⁶ are near the south edge of the mass. The ores are of Triassic or Jurassic age.

Ray, Ariz.—At Ray, 18 m. southwest of Miami, the Pinal schists and diabase are intruded by quartz monzonite porphyry, probably of early Tertiary age. A great blanket of disseminated chalcocite ore is found in the schist, in the diabase, and in the porphyry. The porphyry intrusives near the orebody are small; the mineralized one is less than 0.4 m. in diameter.⁷

Ajo, Ariz.—At Ajo, in the southwestern part of the State, a mass of granite and monzonite, about 4 m. long and 2 m. wide, breaks through rhyolite, tuff, and older gneisses and schists.⁸ Diabases and other dikes intrude the monzonite. The north and west sides of the intrusive are covered with Tertiary and Quaternary rocks later than the monzonite mass; hence the size of the eroded intrusive is not known. The deposits are disseminated copper ores and veins in the monzonite and in the invaded rhyolite. The disseminated ores are in the eastern end of the intrusive and in the invaded rocks, and seem to lie on the axis of a structural ridge of the porphyry. The main orebody occupies an area of 55 acres, has a maximum depth of 600 ft., and contains about 12,000,000 tons of copper carbonate ore and 28,000,000 tons of sulfide ore having a copper content

⁵ S. Paige: Silver City, N. Mex. U. S. Geol. Survey Folio of the Geol. Atlas of the U. S. 199 (1916), 18.

⁶ F. L. Ransome: The Geology and Ore Deposits of the Bisbee Quadrangle, Ariz. U. S. Geol. Survey, *Prof. Paper* 21 (1904) 1–168; Bisbee, Ariz. U. S. Geol. Survey Folio of the Geol. Atlas of the U. S. 112 (1914) 119.

Y. S. Bonillas, J. B. Tenney, and L. Feuchère: Geology of the Warren Mining District. *Trans.* (1916) 55, 284.

⁷ F. L. Ransome: The Copper Deposits of Ray and Miami, Ariz. U. S. Geol. Survey *Prof. Paper* 115 (1919) 192, also Ray, Ariz. U. S. Geol. Survey Folio of the Geol. Atlas of the U. S. 217 (1923) 24.

⁸ I. B. Joralemon: The Ajo Copper-mining District. *Trans.* (1915) 49, 593. A geologic map and section, by Jenkins and Wilson, was kindly lent to the writer by Dr G. M. Butler in advance of publication.

from $\frac{1}{2}$ to 3 per cent. The copper minerals are chiefly chalcopyrite and bornite and, in the oxidized zone, malachite and some chrysocolla. Joralemon states that the sulfide ore is primary, and that there has been little change in it, aside from oxidation. The bottom of the oxidized zone is a nearly level plane, 20 ft. below the deeper gulches and 150 ft. below the hilltops. The orebody, in cross-section, is shaped like a funnel with a flaring top. Lead prospects are opened $1\frac{1}{2}$ m. south of the Cardegan mine and 2 m. south of the main intrusive.

Ely, Nev.—At Ely, monzonite and monzonite porphyry break through Paleozoic limestone and other sedimentary rocks.⁹ The intrusives are probably Jurassic, but may be Tertiary. They are older than the Tertiary rhyolites and tuffs which rest on them. The main porphyry masses form a belt, 7 m. long and from $\frac{1}{2}$ to 1 m. wide, and are broad, stock-like bodies which at places spread out as sills, and seem to be steep-walled cupolas above a roof of a batholith which is largely buried. The disseminated copper deposits are in and near the cupolas, and are absent or scanty in the larger sills. The value of the gold in the disseminated copper ore is from 20 to 40 c. a ton. The gold in the ore is about ten times as valuable as the silver.

Miami, Ariz.—At Miami, the Pinal schists are intruded by granite and quartz monzonite porphyry, probably of early Tertiary age. The mass associated with the ore deposits is about 8 m. long and from 1 to 4 m. wide.¹⁰ The disseminated copper ores lie in the schist and at the edge of the porphyry mass, occupying part of the north border (Fig. 2).

Clifton-Morenci, Ariz.—In the Clifton-Morenci district, a granitic mass, which includes granite porphyry, quartz monzonite porphyry, and quartz diorite porphyry, breaks through Pre-Cambrian, Paleozoic, and Cretaceous sedimentary rocks. Tertiary lavas probably flowed over the eroded surface of the porphyry. It is therefore early Tertiary or late Cretaceous. The belt of intrusive rocks is 11 m. long and 1 or 2 m. wide. Copper ores¹¹ are found on the border of the main intrusive and in dikes that extend from the main mass. The disseminated copper ores are almost free from gold and silver.

Burro Mountains, N. Mex.—In the Burro Mountain district, about 15 m. southwest of Silver City, Grant County, Pre-Cambrian granitic rocks and gneisses are intruded by quartz monzonite and quartz monzonite porphyry, probably of Cretaceous age. The quartz porphyry and

⁹ A. C. Spencer: The Geology and Ore Deposits of Ely, Nev. U. S. Geol. Survey Prof. Paper 96 (1917), 181.

¹⁰ F. L. Ransome: *Op. cit.* (Prof. Paper 115); Geology of the Globe Copper District, Ariz. U. S. Geol. Survey Prof. Paper 12 (1903) 168; Globe, Ariz. U. S. Geol. Survey Prof. Folio of the Geol. Atlas of the U. S. 111 (1905) 17.

¹¹ W. Lindgren: The Copper Deposits of the Clifton-Morenci District, Ariz. U. S. Geol. Survey Prof. Paper 43 (1903) 364.

the quartz monzonite porphyry have been fractured along certain zones, and the fractures are filled and the rocks replaced by chalcocite, forming disseminated deposits¹² enriched by secondary chalcocite. The intruding porphyry mass is about 6 m. long and 3 or 4 m. wide. The deposits are near the northeast rim of the mass.

Silverbell, Ariz.—At Silverbell, 35 m. northwest of Tucson, small bodies of granite and alaskite porphyry intrude limestone, and lean disseminated ores are found in the alaskite porphyry.¹³

South Amole, Ariz.—In the South Amole district, 8 m. southwest of Tucson, a small quartz porphyry mass breaks through sedimentary rocks, and lean disseminated ores¹⁴ occur in the intrusive. The invaded rocks, near the intrusive, contain lead-silver ores.

Copper Creek, Ariz.—In the Bunker Hill or Copper Creek district, between Copper Creek and Mammoth, a town on the San Pedro River, 18 m. southeast of Winkelman, there are low-grade, disseminated copper ores in porphyry.

Cananea, Mexico.—Cananea¹⁵ is in Sonora, 25 m. south of the international boundary and 38 m. by rail from Bisbee. An ancient granite is overlain by Paleozoic(?) quartzites and limestones, and these are intruded by diabase, above which are rhyolite tuffs and andesite. Intrusive masses of syenite and syenite porphyry invade the sedimentary rocks, and larger intrusives of diorite and diorite porphyry cut the tuffs and limestones. The diorites are intruded by granite porphyry, granodiorite, quartz monzonite porphyry, gabbro, and diabase.

The diorite porphyry appears as several bodies, but the largest one—which is southwest of Cananea—is more than 4 m. long and about 2 m. wide. The west edge of this mass is not mapped, but is known to extend west of the area shown in Fig. 3. The diorite porphyry carries some quartz, and is of nearly the same composition as a quartz diorite. This intrusive, where it invades limestone, forms extensive garnet zones

¹² S. Paige: Contributions to Economic Geology: Metalliferous Ore Deposits Near the Burro Mountains, Grant County, N. Mex. U. S. Geol. Survey, *Bull.* 470c (1911), 109–125, 131, 150.

R. E. Somers: Geology of the Burro Mountains Copper District, New Mexico. *Trans.* (1915) 52, 604.

¹³ C. A. Stewart: The Geology and Ore Deposits of the Silverbell Mining District, Arizona. *Trans.* (1912) 43, 240.

C. F. Tolman, Jr.: Copper Deposits of Silverbell, Ariz. *Min. & Sci. Press* (1909) 99, 710.

¹⁴ O. P. Jenkins and E. D. Wilson: A Geological Reconnaissance of the Tucson and Amole Mountains. Univ. of Ariz. *Bull.* 106 (1920).

¹⁵ S. F. Emmons: Cananea Mining District of Mexico. *Econ. Geol.* (1910) 5, 312.

M. J. Elsing: Relation of Outcrops to One at Cananea. *Eng. & Min. Jnl.-Press* (1913) 95, 357.

with associated chalcopyrite orebodies. Such contact metamorphic deposits are found at Puertecitos, Elenita, Elisa and Democrata mines; and parts of the orebodies of the Kirk and West Cobre Grande mines are of contact metamorphic origin. Great deposits of disseminated copper

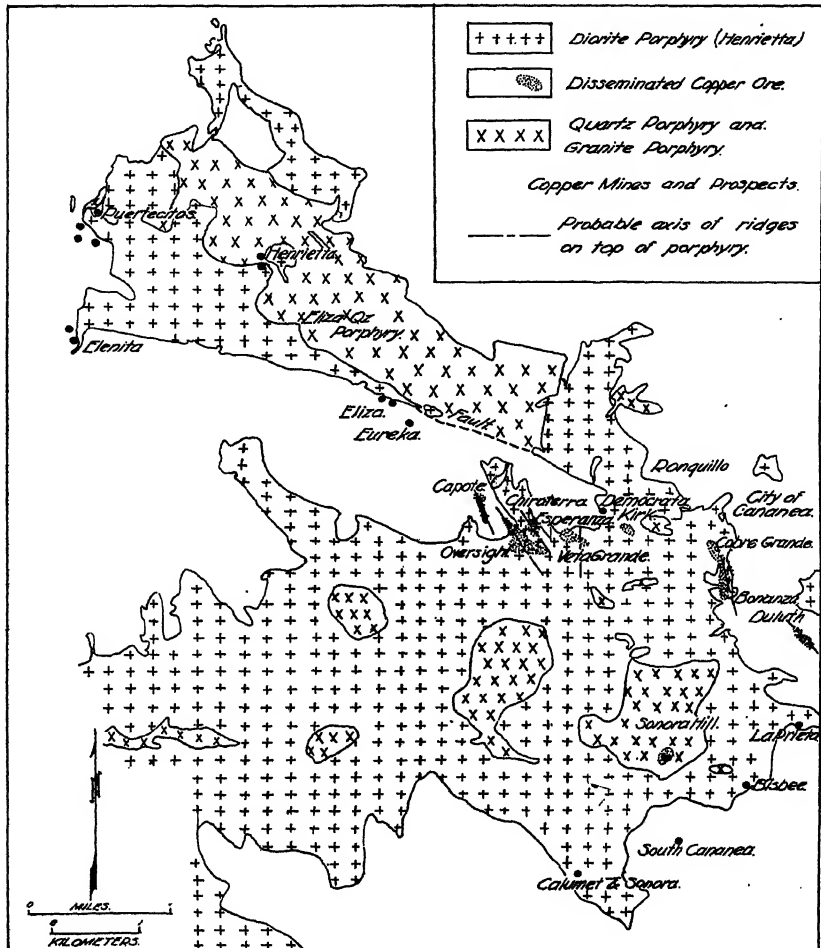


FIG. 3.—MAP OF CANANEA DISTRICT, SONORA, MEXICO, SHOWING RELATIONS OF DISSEMINATED COPPER ORES TO PORPHYRY AND TO FINGERS OF PORPHYRY.

ore are found within the diorite, the chief deposits lying at the eastern edge near Cananea. These include the Capote (in part), the Oversight, Esperanza, Veta Grande, Kirk, Cobre Grande, Cananea Central, Bonanza and Duluth. Disseminated copper ores occur also near the contact of

the diorite porphyry and intruding quartz porphyry on the south slope of Sonora hill, about 1 m. southeast of the summit of Cerro de Cobre. The quartz porphyry, granite porphyry, and granodiorite intrude the diorite porphyry, and probably some metallization attended their intrusion; but the most important deposits are in the diorite porphyry and in the limestone and quartzites where they are invaded by the diorite porphyry. The chief metallization took place soon after its intrusion, although some attended the invasion of quartz porphyry and granodiorite. The disseminated ores occur principally near the edges of the diorite porphyry, and nearly all the chief deposits mined lie on or along the axial planes of ridges of the roof of the intrusive and in part probably were formerly below such ridges at places where the latter have been removed subsequently by erosion.

Iguaran, Mexico.—In the Iguaran district, in the south part of Michoacan, a belt of "granulite," about 2200 ft. wide, is traced several miles on strike. This rock is fractured and seamed with veinlets of copper sulfides, and carries shots of copper ore. There are said to be large bodies of disseminated copper ore carrying 3 per cent. copper.¹⁶

SUMMARY

In the 10 districts shown in Figs. 1 and 2, the intruding rocks, genetically associated with the disseminated copper ores, include granite, granite porphyry, quartz monzonite, quartz monzonite porphyry, monzonite, monzonite porphyry, quartz diorite, quartz diorite porphyry, diorite and diorite porphyry. The most basic rock of the whole series is diorite, which is found in the Morenci and Cananea districts. In the Morenci district the diorite contains 61.1 per cent. SiO_2 . It also contains some quartz, and is on the border line between quartz diorite and diorite. This rock is a basic phase of the metallized porphyry intrusive, and is not closely associated with the ores. The diorite porphyry of Cananea also contains quartz. A noteworthy feature of the intrusives is that they are rarely of uniform composition, but show differentiation, some of which at least has occurred after intrusion. This lack of uniformity is characteristic of small stocks that extend upward from a larger underlying batholith. Diabases, gabbros, and their porphyries are not found as parts of the stocks, and normal diorites are rare.

In age the intrusives range from the pre-Cretaceous to the Tertiary. The Bisbee intrusive is probably Triassic or Jurassic; it is covered with Cretaceous sediments. The Ely intrusives are regarded as probably Jurassic, but similar intrusives of Tertiary age are found in Utah, and the Ely intrusive also may be Tertiary. The Santa Rita intrusive is probably

¹⁶ E. Cumenge: *Bull. Soc. Française de Minéralogie*. (1898) 21, 137.

late Cretaceous. The intrusives of Bingham, Morenci, Burro Mountain (Tyrone), Ray and Miami are all probably Tertiary. The deposits are at or near the tops of cupolas, and seldom occur low on the roofs of batholiths.

Several districts containing disseminated copper ores have been described, of which geologic maps are not generally available. They seem to show features similar to those of the areas noted. At Silverbell lean disseminated ores occur in a small intrusive of alaskite porphyry, and, in the South Amole district, lean ores occur in a small quartz porphyry mass.

The intrusive at Cananea is a diorite porphyry which, as stated, contains some quartz, and is similar in composition to the diorite phase of the Morenci intrusive. At Cananea, also, a rather acid quartz porphyry intrudes the diorite. The ore-bearing intrusive at Cananea is probably one of the largest as well as one of the most basic of all the intrusives in North America containing disseminated copper ore. At Iguaran the ores are in a long narrow mass of "granulite."

The deposits of disseminated copper ores are of the acrobatholithic group, which group, more often than any other, shows zonal arrangements of lodes of the metals. This feature is common in districts containing disseminated copper ores. Zones with zinc, precious metals, or lead-silver commonly are found around the disseminated copper ores. The most extensive zones surround the copper ores of Bingham, Utah, and Ely, Nev., but noteworthy deposits of one or more of the metals named are found in the invaded rocks at Bisbee, Ariz., Santa Rita, N. Mex., Miami, Ray, and Morenci, Ariz., and at Cananea. Lead-silver ores are found also around the disseminated copper ores at Silverbell, Ajo, and at South Amole, Ariz. Deposits of manganese ores are found in the invaded rocks around the copper-bearing intrusives at Ely, Nev., and at Bisbee, Ariz.

The disseminated copper ores seem to be confined to the relatively small intruding masses and to the areas near such masses. The large areas of such rocks, so far as they are known, do not contain them. That is because the deposits are high on the upward extensions or cupolas of the batholiths. The deposits are found in all positions on the outcrops of the cupolas, but they seem to be more plentiful in the rims than in the central areas of the intruding masses, except where such masses are very small, and there they are found in the central parts.

In some districts the disseminated deposits lie along or near the axes of outward-pointing fingers of the intrusive, or along small outlying masses which are probably connected below with the main intrusive and which, by further erosion, will become fingers of the main mass. The deposits as stated belong to the acrobatholithic group. It was noted in

an earlier paper¹⁷ that the most favorable positions for deposits of these groups are at the margins of the intrusives, particularly in and near the outward-pointing fingers of the parent intrusives. The batholiths slope outward, and in general these fingers represent inverted troughs that lean toward the centers of the cupolas. If the country is one of low relief, the boundary of the outcrop of the intruding mass is roughly a contour on its surface. Deeper contours would generally show the leaning ridge farther from the present outcrop, and, before erosion had gone as far as it is now, the trough doubtless extended above the present surface or outcrop of the intrusive. This is illustrated by Figs. 4 and 5.

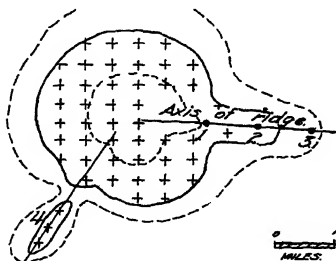


FIG. 4.

FIG. 4.—DIAGRAM SHOWING BY SOLID LINES THE OUTCROP OF A CUPOLA WITH A TONGUE OR RIDGE EXTENDING OUTWARD FROM THE CUPOLA.

The dotted line within the outcrop represents a contour on the roof of the cupola as it was before erosion. The dotted line outside of the outcrop is a contour on the roof below the present surface. Both 1 and 2 are favorable positions for disseminated deposits, and 3 is favorable for deposits outside the cupola. The small outlying mass (4) is also a favorable position.

FIG. 5.—CROSS-SECTION OF A CUPOLA SHOWING A SMALL FINGER POINTING UPWARD FROM THE FLANK OF THE MASS.

Positions 1 and 2, outside the intrusive are favorable; 3, 4 and 5 are favorable positions for disseminated ores inside the intrusive. Point 6 is a less favorable position; points 7, 8 and 9 are unfavorable positions. The top of the cupola appears to be a favorable place for fracturing. The metallizing solutions from the intrusive, moreover, tend to move upward and are deflected to the top of the cupola and to structural ridges near the top, as suggested by arrows.

The distribution of the disseminated ores on the cupolas is controlled largely by fracturing. The ridges or inverted troughs of the cupolas of many batholiths seem to be favorable places for fracturing. The rising solutions that metallized the fractured masses moved upward from the deeper parts of the batholiths to the cupolas, and particularly to the ridges on the cupolas. The axes of the ridges are in general near the axes of the fingers of the outcrops of the intrusive and, consequently, the latter are in preferred positions for metallization.

This relation is well shown in the Cananea district where all the most productive disseminated deposits lie on or very near the axial planes of

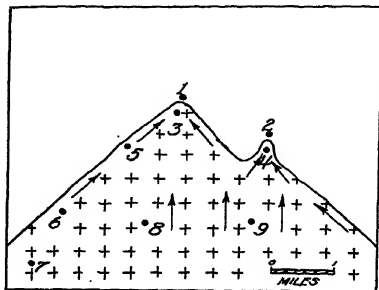


FIG. 5.

¹⁷ W. H. Emmons: *Op. cit.*

projecting fingers of the cupolas, or on outlying masses that probably by further erosion will become fingers. The Duluth orebody is in a small outlying mass of the diorite porphyry. The Capote is in such a mass and in adjacent fractured quartzite and limestone. The Oversight, the greatest disseminated copper deposit in the district, is developed along a finger of crushed diorite porphyry protruding into limestone and quartzite. The Esperanza, Veta Grande and Bonanza-Central are on fingers of the porphyry. This feature is shown also by the deposits at Bisbee, Ajo, Ely, Santa Rita and Tyrone.

Some of the disseminated deposits are well within the main body of the batholith, as shown by the outline of its outcrop. Some of such deposits, nevertheless, lie on the axes of ridges of the cupola, for the ridges probably extended in the general direction of the axes of the part that has not been removed by erosion. In reconstructing the eroded part of the batholith, one is necessarily in the realm of speculation, yet reconstructions may be made that are sufficiently accurate to be of service. There are, of course, many barren axes in the metallized cupolas, and some of the deposits seem to have no relation to recognized axes, yet the relation to the axes is so common that it seems to point to an underlying law. This relation, of course, will hold only in the acrobatholithic and epibatholithic groups of deposits. The other groups seem to have no very definite relation to the shape of the roof, except certain deposits of Group 4, which are in the border region of larger batholiths. These will be treated in a subsequent paper.

ACKNOWLEDGMENTS

The writer wishes to thank T. Evans, general manager of the Cananea Consolidated Copper Co., for a map showing the workings in the Cananea district. These workings have been superposed by the writer on the geologic map made by S. F. Emmons, J. M. Boutwell, L. C. Graton and the writer in 1907; and an abbreviated sketch of this map is shown in Fig. 3. G. M. Butler kindly supplied, in advance of publication, a map of the Ajo district by Messrs. Jenkins and Wilson, data from which were used in making the Ajo sketch, Fig. 1. The ore indications on this sketch are from a map by A. T. Barr and W. C. Lawson supplied through the kindness of M. Curley, general manager of the New Cornelia Copper Co.

DISCUSSION

W. LINDGREN, Cambridge, Mass.—This contribution is valuable, first of all, because it brings before us well-considered generalizations based on the actual detail work done in a great many districts. I think Dr. Emmons is essentially correct in his conclusions and that he proves very definitely that a large part of our ore deposits are derived from igneous intrusions and to a considerable extent from the upper part, the cupolas; but that does not imply that there are not certain other kinds of

deposits which clearly are derived from a much lower horizon, from the lower parts of the batholiths. These are perhaps not so common, not so abundant, but they are in general more dependable and often more valuable than the deposit of this kind, as you will realize, for instance by considering the Billingsley and Grimes discussion of the Boulder batholith. If you go over that paper, you will find that a great many batholithic deposits—similar to, if not exactly acrobatholithic—have been exhausted, and have not yielded the results that might have been expected.

Many of the deposits discussed by Dr. Emmons are simply low-grade material, products of a low-grade mineralization which has happened to be enriched to a considerable degree by descending surface waters and the deposition of chalcocite.

A. M. BATEMAN, New Haven, Conn.—Dr. Emmons has concentrated our attention on the value of drawing generalizations from a great many scattered occurrences, as he has done so admirably here. But I have heard this classification discussed by Dr. Emmons four different times and one point always strikes me; that is, the many new names imposed upon a literature already heavily involved with many names.

G. F. LOUGHLIN, Washington, D. C.—Those interested in details of fracturing might like to see Dr. Emmons give further consideration to the origin of the fractures which he dismisses by merely saying that they were probably due to stresses, presumably through regional deformation. In contrast to that, Locke's¹⁸ recent paper attributed fracturing to the process of "mineralization stoping" solutions rising and corroding the rock, which spalls under pressure around the cavities thus produced. Finally, when corrosion and spalling cease, metalliferous minerals are deposited around the fragments.

I would also like to comment on the valuable contribution to the subject of ore hunting that Dr. Emmons has made by emphasizing the restriction of these disseminated copper deposits to the small cupolas and their tendency to occur in the rims rather than the central part of the cupolas.

H. C. BOYDELL, Cambridge, Mass.—Last summer I spent five months examining what Dr. Emmons referred to as the Burra Mountain deposits, at Tyrone, New Mexico. Emmons says that the deposit tends to be confined to the intruded rather than the intruding rock. That may be true generally, but at Tyrone there are some very important exceptions.

The first discovered deposits centered around Leopold, where mining was first carried on. They were entirely in the monzonite, and one of the most recently discovered deposits at Burra Mountain is also in monzonite. The story of the mineralization there, so far as I was able to elicit it, was that there had been a fracturing earlier than the intrusion and that the quartz monzonite porphyry had followed this, emphasized it and so favored deposition in the pre-Cambrian granite, which is the intruded rock.

I was interested to discover whether, in these intrusions in the granite, the mineralization was as good in the intrusive monzonite as in the granite itself; but this point I could not satisfactorily settle. It seemed to me that of the two the granite was the more mineralized—i. e., contained more protore—and I concluded that it was due to a greater permeability of the granite, which was probably due in its turn to the intruding and wedgelike action of the intrusive itself. I should say that Tyrone at least forms one exception to Dr. Emmons' generalization that the mineralization is in the intruded rather than in the intruding rock.

¹⁸ A. Locke: Formation of Certain Orebodies by Mineralization Stopping. *Econ. Geol.* (1926) 21, 431-453.

D. H. McLAUGHLIN, Cambridge, Mass.—I confess I choked a little over the new names, but I am willing to try to use them, for I feel that Dr. Emmons has made a valuable contribution, particularly in presenting an effective summary of the evidence bearing on the distribution of orebodies about intrusive masses. The generalization that apically truncated stocks are favored regions for ore deposition receives strong support from his work and thereby gains and deserves even firmer standing among geologists.

I have observed certain relations between ore deposits and intrusive masses in the Andes that may perhaps be worth mentioning as supplementary evidence supporting Dr. Emmons' results. The ore deposits of central Peru are closely associated with small stocks which outcrop along the crest of the range and in the high country to the east. The stocks are presumably outliers of the larger batholithic mass which lies along the lower western slopes of the range and reaches the coast in some places. The batholith itself rarely shows indications of mineralization and is justly regarded as unpromising territory for exploration, whereas the zone to the east has long been noted for its mineral deposits. It is noteworthy, that in nearly every case, the stocks that are accompanied by important mineralization are of small area of outcrop. For example, the area of quartz-monzonite-porphyry, from which the orebodies at Cerro de Pasco were presumably derived, is not over a mile and a half in diameter, and the intrusions responsible for the mineralization in other districts seem to be of comparable size. The generalizations brought out in this paper seem to be capable of use in a very definite way in guiding the mining engineer or ore-hunting geologist away from the barren heart of the batholith and to the more promising zone of small outliers.

I regard it of greatest importance to study in the closest possible ways all details of form of intrusions and the relations that ore deposits show to particular promontories and flutings on their sides. Undoubtedly, channelways and zones of permeability are determined to an important degree by such features and exact information concerning them is of practical value in guiding exploration to places most favorable for ore depositions; but it is rather dangerous to draw conclusions about the form of intrusive masses simply from examination of the outlines of the horizontal section exposed on the surface. I entirely agree with Dr. Emmons that the irregularities in form of the intrusive bodies may be of great significance, but, before accepting the evidence as conclusive in any particular case, we must have more data than merely the outline of the intrusive mass on the surface.

G. H. GARREY, Philadelphia, Pa.—Practically all of the deposits that I am familiar with could be grouped under the classifications Dr. Emmons has brought out in his paper, but I also think that there is danger in depending too much on the outline of the outcrop. An outcrop can be covered over easily by a very shallow amount of other formations and one may thus gain a very wrong impression of the true shape of the main intrusive body from the outcrop. One should in any case keep this point in mind when he is trying to determine the contour of the igneous mass and to locate the most favorable places for ore occurrence in any particular district.

I also was impressed by the coinage of a large number of names which a person in general practice would not remember for long and would probably prefer not to remember. I think it is sufficient to know the particular position in which the ore occurs with reference to an intrusive mass, without attaching such names as acro-batholithic. I suppose these names do probably serve a useful purpose, but their use will be pretty nearly limited to philological purposes. I do not care what a man calls a rock or an ore occurrence so long as he knows the relation of the ore to that rock in the particular camp in which he is working. I think the main idea is to know where the ore occurs.

W. LINDGREN.—I do not think it desirable to interfere with the urge to create new names. The thing will be remedied automatically. It will be accepted or it will not be accepted.

R. J. COLONY, New York.—I note the emphasis placed by Dr. Emmons on the necessity of fractured ground for ore deposits. I have been very much impressed myself with the ability of things to get through rock without any fracture at all, so that in many places one finds rock thoroughly permeated and impregnated with certain kinds of mineral deposits where no fracturing whatever has occurred; that is, no fracturing commensurate with the deposit. I suppose those deposits are situated in the deeper places in the batholith; that is, within the batholith itself. I have in mind more particularly some of the magnetite iron-ore deposits in southeastern New York, where there is none but post-ore fracturing, and where the material has penetrated and permeated without any difficulty at all.

Another striking thing brought out by Dr. Emmons, is the way in which all of the deposits described tie up so nicely with the theory of zonal distribution.

J. L. DYNAN, Tonopah, Nev.—I have spent short periods in copper camps, which come under the classification described by Dr. Emmons, and so far as my limited observation goes, I am in hearty accord with what he has said.

A. M. BATEMAN.—I have spent quite a little time in all of the camps that are mentioned here, and in regard to the point that Professor Colony has brought up, that of fracturing, the thing that perhaps has impressed me most in all of the so-called disseminated porphyry copper camps throughout the country is that fracturing is such an important feature. I think the word disseminated is oftentimes wrongly used to describe the occurrence of ore as it is found, for that term usually implies the pepper and salt effect of distributed grains of sulfide within the rock itself; whereas, as one walks through mile after mile of workings and bench after bench of open-cut workings in the various disseminated or porphyry copper camps, what strikes one is the great amount of fracturing so that the ore consists not only of disseminated grains but also of small interlacing veinlets of quartz or of sulfide, or of quartz and sulfide together.

H. C. BOYDELL, Cambridge, Mass.—I would like to raise the point whether the structural feature is not an essential preliminary to all ore deposits. The emphasis on the cracks and the fractures here is merely reference to what I consider the essential structural features without which there would be no ore deposition at all, for the simple reason that the solutions would have had no channel of access.

R. J. COLONY.—I think that I can state definitely that there are occurrences where the ores were deposited without any great extensive fracturing. In the magnetite deposits of New York State there is fracturing, true, but the greater part of it is post-ore fracturing—that is, post-ore deformation—and the ore seems to have had the capacity to penetrate in a remarkable way. These are the deposits that I have called "magmatic" replacement deposits; they are undoubtedly connected with the end-stage process of the consolidation of an igneous mass that has attacked rock, which in this case was previously metamorphosed, intensely modified and folded. The ore has followed those particular zones or beds which were calcareous, effecting complete replacement of those beds without any extensive fracturing at all.

H. C. BOYDELL.—That is merely a case of greater solubility which might in part be induced by structure.

W. H. GRANT, San Francisco, Cal.—You probably have heard a good many times from people outside of our profession about things that we have not discovered

within it. Several years ago a division chief of the supervising architects' office in Washington asked me if I could find any building stone west of the middle western plains which could be used in governmental buildings, because specimens that had been sent to them from that western territory had weaknesses, due to what the architect calls "dries"—microscopic cracks. The samples did not pass the strength test that architects require for building stone.

That was a rather sweeping statement to me—that all rocks in the western United States contain "dries" and were not suitable for building stones. There would probably be some exceptions, at least in small quantities, but it indicates that there are a great many cracks in rocks which a field geologist will not recognize with the naked eye.

E. F. BURCHARD, Washington, D. C.—Undoubtedly, on a large scale, we do not recognize all such fractures without the aid of the microscope. I have had some difficulty also in seeking out material for the supervising architect's office. That was particularly true of granites and marbles of the West. I am wondering whether a different history of the Adirondack region, involving much earlier folding and fracturing, may not have a bearing on this question and if some of the cracks which may have existed have become recemented or welded together.

H. RIES, Ithaca, N. Y.—A permeability test sometimes made on building stones consists of soaking the rock in an alcohol solution of nigrosine. It is remarkable to see to what distance the alcohol coloring agent will penetrate even a slightly porous rock; which, of course, shows its permeability to liquids that might stain it. A rock which appears to be exceedingly dense and shows a very small absorption when soaked in water may be penetrated by the nigrosine solution to a depth of $\frac{1}{2}$ to $\frac{3}{4}$ in. in 24 hr. I mention this fact to show that some apparently dense rocks are appreciably permeable and that, because of this fact, ore-bearing solutions might occasionally penetrate rocks for some distance even though the latter were free from megascopic fractures.

F. L. RANSOME, Tucson, Ariz. (written discussion).—The title of Dr. Emmons' very interesting paper is, I think, slightly misleading, inasmuch as some of the most important deposits of the group, such as those at Ray and Miami, Ariz., are preponderantly in schist. At Miami, much of the ore is in schist, which is beneath the porphyry that is exposed at the surface. I venture to offer the suggestion that the title might read "Relations of Disseminated Copper Ores to Igneous Intrusives."

The proposed terminology of the classification is decidedly formidable and this is likely to be some deterrent to its use.

Dr. Emmons' generalization that the disseminated copper deposits appear to be confined to relatively small masses of the intrusive rock and to areas near such masses is true, so far as my experience goes, although it has not hitherto, so far as I am aware, been definitely stated. The recognition of this relationship between the form of the igneous mass and the occurrence of an ore deposit is not only of theoretical importance but likely to prove of great utility. It is conceivable that under certain conditions, by the application of this generalization, copper deposits that are completely concealed under younger detrital deposits or lava flows may be brought to light.

W. H. EMMONS (author's reply to discussion).—I regret that I was unable to attend the meeting at which my paper was presented, and to take part in the discussion. I wish to thank Mr. Garrey for reading my paper, and the members present for the thoughtful consideration which they have given it.

I agree with Professor Lindgren that many valuable mineral deposits are situated low on the batholiths. These, however, are low on the roofs and are almost never

found within, far from the contacts. Deposits of gold greatly predominate in such deposits.

The lodes of Butte, near the rim of the Boulder Batholith, which was described by Billingsley and Grimes, are pretty clearly of later age than the main intrusive. They are definitely zoned¹⁹ around the center of quartz porphyry intrusives that cut the granite. The zones from the center out show the normal order: (1) enargite, (2) enargite with considerable sphalerite and silver ores and some rhodonite and rhodochrosite, (3) zinc and manganese and silver minerals with little copper.

The second largest group of deposits associated with the Boulder Batholith consists of the silver-lead deposits of the Alta region, described by Knopf. These are grouped in and near a roof pendant well toward the center of the batholith and are probably the greatest group of lead-silver ores in the world that are associated with such a roof pendant. Except Butte, the deposits near the border of the batholith show little or no horizontal zoning. This is commonly the case around batholiths as large as the Boulder Batholith.

Small unprofitable deposits, as Professor Lindgren states, are not uncommonly found around the cupolas. Yet the same may be said of the deeper seated deposits. Inspection of maps of endobatholithic regions such as the Canadian Shield, Southern Rhodesia, or Western Australia, will show that there are scores of small and as yet unsuccessful developments in the regions of the great gold bonanzas. The position of the deposit on the batholith is only one feature of its genesis. Where valuable deposits are found, suitable fracturing and other favorable conditions must be provided and in certain surroundings the conditions for enrichment also must be favorable.

Professor Bateman and others also are not impressed with the need for the terms which I have proposed for a classification of epigenetic ores. I have merely used Greek prefixes, most of which are already in rather general use in the geological literature. If one remembers this, the words will seem less formidable. If they are useful, they will be used; if they are not, they will be forgotten. That evidently would please Mr. Garrey. I may add here that the classification is not particularly necessary or helpful when a few deposits only are treated and compared; but where a large number are studied it is useful.

Replying to Dr. Loughlin and to Professor Colony, I believe that the localization of essentially all epigenetic ore deposits depends upon fracturing. That holds, in the main, even for contact metamorphic deposits although the fracturing is often masked in the latter. As everyone knows, the copper ores of the porphyry group are in rocks that are intensely shattered and fissured. That is one of the most conspicuous features of the deposits. In some of the districts described, it probably is not possible to trim a specimen of the metal-bearing porphyry without breaking it along planes of early fracturing. Even the parts of the porphyry that are not mineralized with workable ore are commonly shattered and fractured. However favorable other conditions, the ores are not deposited unless conditions of fracturing are favorable also.

Replying to Mr. Boyde, I wish to note that the disseminated copper ores are found in the intruding rock in every district mapped (Figs. 1 and 2). In a few of them the ores are found also in the invaded rock. Contrary to the inference of Mr. Boyde, I have not made the generalization that the mineralization is in the intruded rather than the intruding rock. That statement would apply to Group 5 (endobatholithic deposits), but not to Group 2 (acrobatholithic deposits), to which all of the deposits of disseminated copper ore belong.

Professor McLaughlin's statement regarding the small ore-bearing intrusives of Peru is of great interest. We have been hoping that some of the geologists working in that region might publish at least a general map showing the relations of the

¹⁹ R. H. Sales: *Trans.* (1913) 46, 3.

deposits to the intrusives, but thus far none has appeared. Evidently the general relations in Peru are much like those in the Bolivia Plateau region although in Bolivia the evidence of normal zoning about the small intrusives is much more clearly presented.

I am very much interested in what Professor Colony said about the magnetite ores of southeastern New York which seem to have no relation to fracturing. Essentially all of the epigenetic ores with which I am familiar have a definite relation to fractures and in general that is a very prominent feature of their occurrence. Even bedding-plane deposits that replace limestone may often be followed into the original limestone beds when the latter are seen to be fractured by bedding-plane fissures.

The classification of ore deposits with respect to their positions on the batholith, the studies of the contour of the roofs of batholiths and the studies of zones around cupolas do not serve to locate deposits. These studies merely outline favorable places for search and development.

Geology of Pioche, Nevada, and Vicinity*

By L. G. WESTGATE,† DELAWARE, OHIO, AND ADOLPH KNOPF,‡ NEW HAVEN, CONN.

(New York Meeting, February, 1927)

PART I.—AREAL GEOLOGY§

PIOCHE lies 240 miles southwest of Salt Lake City, in southeastern Nevada, 19 miles west of the Nevada-Utah line. It is at the end of a branch line (33 miles), which connects at Caliente with the Los Angeles and Salt Lake division of the Union Pacific System. Its commercial relations are with Salt Lake City and Los Angeles, chiefly with the former.

The surrounding area is part of the Great Basin, an arid region of alternating mountain range and intermont plain, the latter floored with outwash from the mountains. About Pioche the plain varies between 4700 and 6000 ft. above sea level. The town lies in a valley at the east base of a low northwest-southeast range (the Pioche Hills) 12 miles in length, which rises about 1000 ft. above the adjacent flat. West of the Pioche Hills is a higher north-south range consisting of two parts: to the south the Highland Range, between Bennett Pass and Stampede Gap; to the north the Bristol Range, from Stampede Gap to Bristol Pass. This double range reaches altitudes of 8000 to 9000 ft., or more (the highest point, Highland Peak, is 9395 ft.), and rises some 2500 ft. above the surrounding plain. It is continued in lower hills to north and south. Smaller ranges rise from the flat west of the Highland and Bristol ranges. The area studied and shown on the map (Fig. 1) is covered by three inch-to-the-mile topographic sheets: the Highland Range, the Bristol Range, and the Panaca, the latter mapped in part only.

The sedimentary rocks of the district include Paleozoic limestones, sandstones and shales, Pliocene tuffs, and modern wash. Igneous rocks are represented by intrusive dikes and stocks and by extensive lava flows of mid-Tertiary time.

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† Professor of Geology, Ohio Wesleyan University; Assoc. Geol. U. S. Geological Survey.

‡ Professor of Geology, Yale University; Geologist, U. S. Geological Survey.

§ By L. G. Westgate.

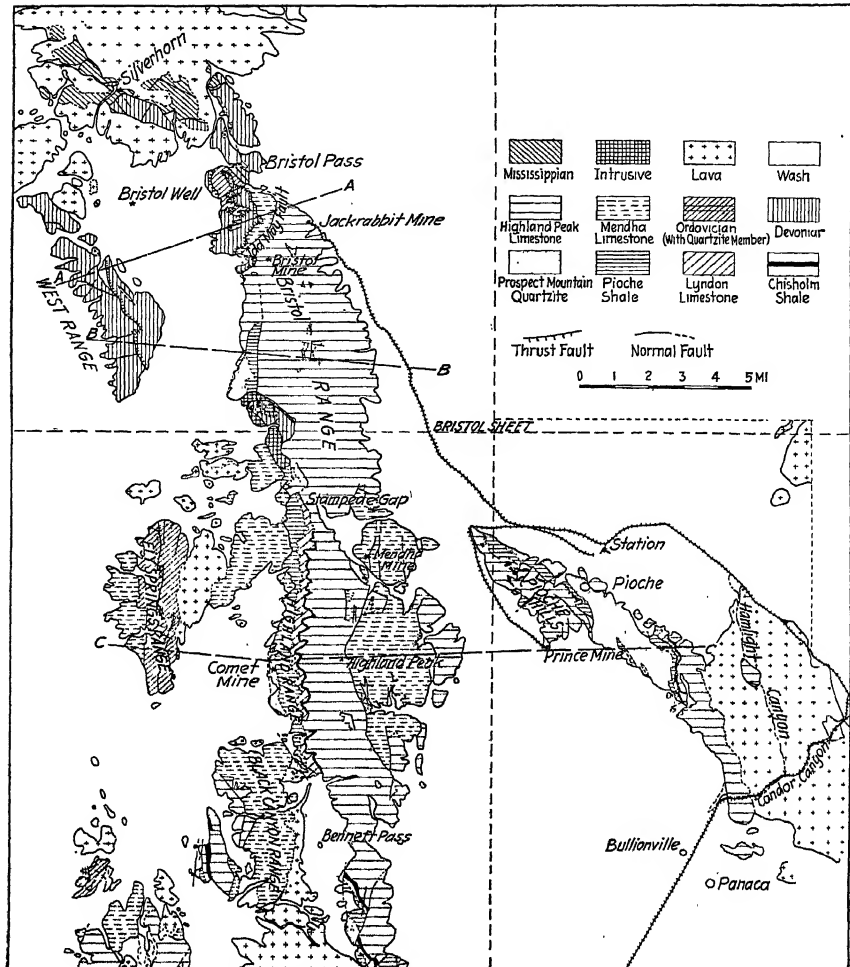


FIG. 1.—GEOLOGICAL MAP OF THE PIOCHE DISTRICT

SEDIMENTARY ROCKS

PALEOZOIC FORMATIONS

The Paleozoic formations, nearly 18,000 ft. thick, are shown in the columnar section (Fig. 2). Numerous faults have made it impossible to get continuous sections, and the uniform character of many of the formations has made it difficult or impossible to piece together partial sections to a satisfactory whole, so that too great reliance can not be placed on the figures for thickness, especially for the thicker formations.

Cambrian sediments only are found in the Pioche Hills and the Highland Range and, except in insignificant amounts, in the Bristol Range south of the Ida May fault, which crosses the range in a north-east direction north of the Bristol mine. No production has come so far from any except Cambrian formations. Detailed descriptions will therefore be given only for the Cambrian.

CAMBRIAN

Prospect Mountain Quartzite.—This formation, which is named from its type exposure in the Eureka district, Nev., is a quartzite, red where weathered, usually showing sand grains under the hand lens. Cross-bedding, lines of small pebbles, and, rarely, ripple marks have been seen. The beds range in thickness from 1 to 5 ft. and in the upper part are massive ledges. The partings are generally shaly layers which resemble the overlying Pioche shale. In places shale bands several feet thick occur in the quartzite. At the Comet mine, 10 ft. of this shale is separated by some 25 ft. of quartzite from the overlying Pioche. This interbedding of the shale in the upper part of the quartzite sometimes makes it hard to decide whether one is dealing with the Pioche shale or the Prospect Mountain quartzite.

The Prospect Mountain quartzite is the lowest known formation in the Pioche district. The deeper shafts at Pioche were still in it at their greatest depth, at the 1400-ft. level. As they start below the top of the formation, the thickness of the quartzite is certainly 1500 ft. and may be much more. Its total thickness and the character of the underlying rocks in the Pioche district are unknown.

The largest exposure of the quartzite crosses the Pioche Hills obliquely and forms the hills north and south of Pioche. It was from veins in the quartzite near Pioche that the chief production of the district came in the early days. The Prospect Mountain quartzite is also found in places along the west base of the Highland and Bristol ranges to four miles north of Stampede Gap.

Pioche Shale.—The Pioche shale is 1120 ft. thick along the west base of the Highland Range, and underground workings near Pioche seem to point to a similar thickness. In the Highland Range much of it is a yellow-brown or red fine-grained arenaceous shale, very finely micaceous.

MISSISSIPPIAN	MISSISSIPPIAN		700	Gray rather heavy bedded limestone. Fossiliferous Break
			2275	Gray medium-bedded limestone, some layers cherty. Fossiliferous Basal beds weathering yellow and rusty-brown. Very fossiliferous
			700	Red thick or thin-bedded quartzitic sandstones. Some shaly sandstones
			500	Mainly coal-black aphanitic limestones, weathering gray or white. Some bands rusty-weathering. Some shaly layers.
			300	Break Gray limestones, slightly cherty. Fossils.
DEVONIAN	Upper	West Range	550	Blue-gray, medium and heavy-bedded limestone. Medium-bedded, partly nodular limestones, weathering to yellowish debris. Fossils
	Middle	Silverhorn	3000	Gray limestones and dolomites alternately medium or thin-bedded. Beds of brown "spaghetti" (coralline) dolomite conspicuous in middle and upper part of formation. Break SILURIAN Gray and brown dolomites 15'
			675	Dark thin-to thick-bedded limestones and dolomites, many layers cherty. Fossiliferous.
ORDOVICIAN			450	White to pink quartzite member Gray limestones, thin bedded, but often making thick ledges. Fossiliferous Gray limestones, some conglomeratic. Fossiliferous
			670	Break Gray limestones, weathering to gray or yellowish slabs or debris. Fossiliferous Break
			450	Blue-gray, thick or thin limestone, somewhat cherty. Layers of flat-limestone conglomerate
CAMBRIAN	Upper	MENDOTA	350	Dark gray to black dolomitic limestone. Cherty.
			300	Gray heavy-bedded crystalline limestone.
			800	Gray limestones, thin to thick-bedded. Much oolitic or conglomeratic material.
	Middle	HIGHLAND PEAK	3000	Heavy ledges of light colored limestone near top of formation, making cliffs along crest of Bristol Range. Laminated white dolomitic limestone to 50' interbedded in middle and upper part of section. Light to dark gray limestones, heavier below, often thin-bedded and slabby above. ----- In Pioche Hills, rusty-weathering siliceous dolomite overlain by cherty limestone. Total 100', about 1200' above base of formation.
			150	CHISHOLM - Brownish-yellow to red argillaceous shale. Fossils abundant
			400	Heavy-bedded limestone. Upper half light, lower dark.
	Lower	PIOCHE MOUNTAIN	1120	Yellow-brown or red micaceous shales with frequent interbedded limestones or occasional interbedded sandstones.
			1500	Light to dark-red vitreous quartzitic sandstone. Often interbedded with shale partings. Heavy ledges at top.

FIG. 2.—COLUMNAR SECTION.

In some places it is coarser and more micaceous, approaching a schist in appearance. The upper beds at Pioche are of this character; a gray-banded schistose rock showing under the microscope small grains of quartz, flakes of chlorite, muscovite and biotite, and a considerable amount of calcite. The mica flakes reach $\frac{1}{2}$ mm.; the quartz grains average $\frac{1}{25}$ mm.

Layers of gray limestone from 2 to 15 ft. thick, massive, laminated or nodular, and, less commonly, layers of micaceous or quartzitic sandstone, are interbedded in the shale. Twelve limestone and three sandstone layers were seen in the Highland Range section. In places the limestones have been replaced by sulfides. The Combined Metals mine at Pioche is working such a deposit in a layer some 250 ft. above the base of the formation.

Fossils are common in the limestone layers and are found, though rarely, in the shale. At one point 400 ft. above the base of the shale near the Comet mine abundant trilobites were found, but this is exceptional. Walcott¹ places the Pioche shale in the Lower Cambrian.

The Pioche shale occurs at various places in the Pioche Hills and at intervals along the west side of the Highland-Bristol Range from the south edge of the mapped area to within 3 miles of the Bristol mine.

Lyndon Limestone.—The Lyndon limestone, 400 ft. thick, lies between the Pioche shale and the overlying Chisholm shale. The lower 200 ft. is a dark gray, rather heavy-bedded limestone; the upper part a light gray to white and more crystalline limestone. Where the Chisholm is present the Lyndon can be mapped separately from the Highland Peak. Where it is absent or where the two limestones are brought against each other by faulting they are so much alike that it is not possible to map them separately; in such places some Lyndon may have been included in the areas mapped as Highland Peak. This may be the case north of Stampede Gap, where the Chisholm was not seen, and in places in the Pioche Hills.

Chisholm Shale.—The Chisholm shale, named by Walcott in 1916 from the Chisholm mine 2 miles west of Pioche, is a fine-grained, yellow-brown to red laminated argillaceous shale which weathers to small flat débris. It is abundantly fossiliferous and was placed by Walcott² in the Middle Cambrian. Like the Pioche, it includes interbedded limestones. The Chisholm can be told from the Pioche by the absence of mica, by the almost universal presence of fossils, especially fragments of trilobites, and by its relation to the adjacent Lyndon and Highland Peak limestones. It is the most sharply marked formation of the Cambrian and proved indispensable in working out the structure of the Pioche Hills and of parts of the Highland Range.

¹ C. D. Walcott: Cambrian Brachiopoda. *Monograph* 51, U. S. Geol. Surv. (1912) Pt. I, 158.

² C. D. Walcott: *Op. cit.*, 192.

Highland Peak Limestone.—The formation to which the name Highland Peak limestone is given is the thickest formation in the district and occupies the greatest area. It makes the Highland-Bristol Range, except for the unfaulted mass of Mendha limestone, as far north as the Ida May fault. In the Pioche Hills no higher beds than the middle Highland Peak occur. No unfaulted section has been found and 3000 ft. is as close an estimate as can be given. It may be considerably too small.

The rock is a gray to black limestone, though many of the layers are dolomitic. The beds reach 4 to 5 ft. in thickness although many parts are thin-bedded. Even these thin-bedded parts often make ledges of appreciable height which weather to slabby *débris*. Layers are often variously color-mottled on cross-fracture, due to differences in coarseness of grain or in the extent of dolomitization. The whole formation is a repetition of these variations, so that it is difficult to tell where one is vertically in the Highland Peak limestone by the character of its common rock types.

Three exceptional rock types occur which help one to determine the vertical position in a general way. These are:

1. Gray, siliceous, laminated dolomites and dolomitic limestones from 2 or 3 to more than 60 ft. thick which weather light gray or white. In many of the beds the laminae are broken, making a sort of flat-pebble conglomerate, though the fragments are usually angular and not rounded. These beds are found in the middle and upper part of the Highland Peak and are not found in any other Paleozoic formation.

2. Thick ledges of well- and heavy-bedded limestone, often white, the parting planes, to a foot in thickness, made of laminated limestone like the type just described. They are found near the top of the formation. They are too high to be found in the Pioche Hills but are conspicuous features along the crest of Bristol Range.

3. East of Panaca in the Pioche Hills a striking combination of beds was seen, from the top down as follows:

- e. Thin-bedded black limestones passing up into heavier-bedded, rough-weathering, finely mottled beds; the normal Highland Peak.

- d. Gray, thin-bedded laminated limestones with large masses of black chert and some white chert in the lowest layers; 17 feet.

- c. Gray siliceous dolomite weathering deep rusty brown; thin-bedded above giving platy *débris*; thicker below weathering to chunky pieces; 40 feet.

- b. Dark to black sub-laminated limestone, 18 feet.

- a. Heavy gray limestone specked with calcite in beds to 6 ft., weathering light gray. Some layers at the top are laminated. These beds are in places dark and may have been so generally at the first; 23 feet.

The combination of beds *a*, *c* and *d* makes a horizon which can be recognized with certainty. Four miles southeast of Pioche it lies 1200

ft. above the Chisholm shale, but this measurement was taken where faulting probably makes exact measurement impossible. The horizon can be traced north from Panaca for several miles and is again found in the hills west of Pioche, north of the Prince mine, where it can be used to work out the faulting. It was not seen in any of the traverses of the west side of the Highland-Bristol Range and probably does not occur there; though it was found at one point at the base of the range near the Jack-rabbit mine.

Mendha Limestone.—The formation to which the name Mendha limestone is here applied shows in ascending order (Fig. 2) thin to thick-bedded oölitic limestones, dark and light dolomites, in part cherty, and thin-bedded blue-gray limestones with cherty layers. The Mendha is the only part of the Cambrian that is in any degree cherty. The basal beds, by contrast with the heavy underlying Highland Peak, make an easily recognized horizon which has helped in working out the structure of the main range. These beds are thin-bedded dark-gray limestones, crystalline and oölitic, and abundantly fossiliferous, which weather red or yellow.

The Mendha is not found in the Pioche Hills. It occurs in the Highland Range as a down-faulted area making the larger part of the east slope of the range, and extends north to include Arizona Peak and a little of the south end of the Bristol Range. It reappears north of the Ida May fault on the east side of the range and is also found in the lower ranges west of the Highland Range. The mines in the Arizona Peak area are in the Mendha. The formation is named for its exposures at the Mendha mine 6 miles west of Pioche. It is considered to be of upper Cambrian age.

ORDOVICIAN AND SILURIAN

The columnar section shows nearly 2000 ft. of limestones, dolomites, and some included quartzites belonging, except for 75 ft. of Silurian at the top, to the Ordovician. These are well shown in the Ely Springs Range on the Highland sheet. They are found in the main range only between the Ida May fault and Bristol Pass, beneath the overthrust fault. No mining has been done in them and they will not be further considered.

DEVONIAN

Silverhorn Limestone.—The Silverhorn limestone comprises some 3000 ft. of gray and brown thick and thin-bedded limestones and dolomites, with quartzites at the top. A characteristic facies is a brown dolomite, called in the field "spaghetti limestone," crowded with small white curving rods formed by the replacement of the small branching corals.

West Range Limestone.—The quartzites at the top of the Silverhorn are overlain by several hundred feet of thin and medium-bedded limestones, certain layers of which are nodular and yellow weathering. At several levels abundant fossils are found.

The two Devonian formations make the whole of the West Range, which lies directly west of the Bristol Range. The Silverhorn constitutes the rock of the overthrust block at the north end of the Bristol Range and both are well shown north of Bristol Pass. No producing mines have so far been developed in them.

MISSISSIPPIAN AND PENNSYLVANIAN

Several thousand feet (Fig. 2) of Mississippian limestones and quartzites occur, almost wholly in the area northwest of Bristol Pass. One isolated hill shows some 700 ft. of limestones, from which Dr. Girty identified Pennsylvanian fossils. These formations are outside any of the mining areas and do not need further consideration here.

TERTIARY

PANACA FORMATION

Meadow Valley Wash and its tributaries, which include the Patterson Wash east of Pioche, drain to the Colorado River. They have cut several hundred feet below the original wash covering of the valley fill and have exposed a thick series of white, buff and red, bedded tuffs which are well shown in the picturesque bad lands east and west of Panaca, and less clearly up the valley of Patterson Wash to well north of Pioche. They vary from a fine silt to sand. Thin bands of fragments to over an inch in diameter sometimes occur. The material varies from an incoherent powder to firm rock. It is composed of mineral grains of volcanic origin, of fragments of glass, clayey material, and carbonate aggregates. Half of the samples showed the presence of diatoms, a few in sufficient abundance to be a diatomaceous earth. The beds prove that there was a great discharge of volcanic dust from some center now unknown, which was washed into the valley, filling it to a depth of several hundred feet. The fill was deep enough to cover the southern end of the Pioche Hills, and the streams developed their courses on the surface of the fill, without regard to the buried topography. When later the streams cut into the fill, they developed canyons (Hamlight and Condor) where they crossed the buried ridges. Probably the canyon above Caliente, followed by the railroad to Pioche, is due to the same cause. The edge of Condor Canyon stands some 700 ft. above the level of the adjacent valley and this gives a minimum measure of the depth of the fill.

Stock³ states that certain mammal fossils found in the tuffs near Panaca "suggest that the deposits are of Pliocene age." Of the diatoms, Dr. Albert Mann, diatomist of the Carnegie Institution, says that "a little over half the species here recorded are abundant in the Alabama

³ Chester Stock: Late Cenozoic Mammalian Remains from the Meadow Valley Region of Southeastern Nevada. *Bull. G. S. A.* 32, 146.

fossil deposit, which has usually been assigned to the Pliocene. I do not know that they may not be equally referred to the Pliocene." The tuffs are in this paper tentatively referred to the Pliocene, chiefly because of the mammalian fossils.

MODERN WASH

Modern wash floors the valleys. Dry Lake Valley west of the Highland-Bristol Range is without outside drainage and shows only the surface of the modern wash. East of the Highland-Bristol Range, in the basin of Meadow Valley and its tributaries, the original coarse gravelly wash which covered the Panaca formation has been redistributed at successively lower levels and now makes a thin veneer over the surface of the flats cut in the Panaca formation or, as slide material, covers their front slopes.

IGNEOUS ROCKS

QUARTZ DIORITE AND ALLIED ROCKS

North of Stampede Gap, in the west base of the Bristol Range, a complex of igneous rocks is found. This includes on the south and west massive coarse-grained rock, generally a quartz diorite. Farther north are dikes and larger masses of massive to porphyritic dioritic rocks. Associated with these are large masses of flow breccias. The whole series is associated with an unfaulked block of Devonian. These rocks need further petrographic study; but as they are not associated with any producing mines, their description is not necessary.

The limestone is changed to a white marble for several hundred feet east of this igneous series, and near the contact contains much yellow garnet and unusual contact minerals at certain points. The garnet, iron and other minerals at the "iron blow-out" a mile south of Stampede Gap may have come from the subsurface extension of this igneous series.

DIKES

Two large basic dike-like masses cut the Prospect Mountain quartzite at the south end of the Pioche Hills.

Porphyry dikes occur, but their importance in the ore geology of the region has been greatly overestimated. A few may have directed the course of ore-bearing solutions, but they have not made any direct contribution of material to the veins. It is not unlikely that the dikes, the larger igneous intrusions, and the ore deposits may all have a common source in some deep-lying unknown body of igneous rock. If so, the ores have a general and indirect, not a direct and specific relation to the dikes.

LAVAS

Large areas are underlain by lavas. The best exposure is in Condor Canyon, where 6000 ft. of eastward-dipping lavas, tuffs, and breccias were measured. Sixteen flows were seen (there may easily be more), the largest being 800 ft. thick. The rocks are chiefly of intermediate types, dacites and andesites, with a little rhyolite. On the Highland and Bristol quadrangles the lavas are similar, the commonest variety being a red dacite, but they weather easily and good sections are not common.

There is nothing within the Pioche district to date the lavas. They are assumed to be, like the lavas generally of the Great Basin, mid-Tertiary, probably Miocene. Before the deposition of the Panaca formation at the end of the Pliocene they had been tilted and faulted, and eroded to essentially the present condition.

STRUCTURES

FAULTING

Faulting is general and both thrust and normal faults occur (see Fig. 3). The beds are often gently bent, probably incidental to the faulting, but sharp folding was seen at but one or two places and there only as single folds.

THRUST FAULTING

A thrust fault, which leaves Devonian above Devonian and earlier rocks, is found at the north end of the Bristol Range (see map, Fig. 1), along the east side of the West Range, and along the west side of the Highland-Bristol Range near Stampede Gap. The evidence for the faulting is the low dip of the fault plane and the almost complete brecciation of the overthrust block. The push seems to have come from the east. In the West Range the fault block overrides the lavas and so is post-Miocene.

NORMAL FAULTING

Normal faulting is widespread. The larger faults are shown on the map but wherever detailed mapping is done, as at the Prince, Combined Metals, or Bristol mines, a large number of small faults are found which only a large-scale map could show.

Normal faulting took place both before and after the thrust faulting. In the West Range ENE-WSW faults come up to but do not affect the thrust fault and the Ida May fault does not displace the thrust fault in the main range. These faults, together with a general northwest dip from here north seem to be earlier structural features.

It is also true that the overthrust block in the Bristol Range is itself faulted. Just south of Bristol Pass it is cut by NE-SW faults and east of this a NW-SE fault drops the overthrust block to the level of the valley.

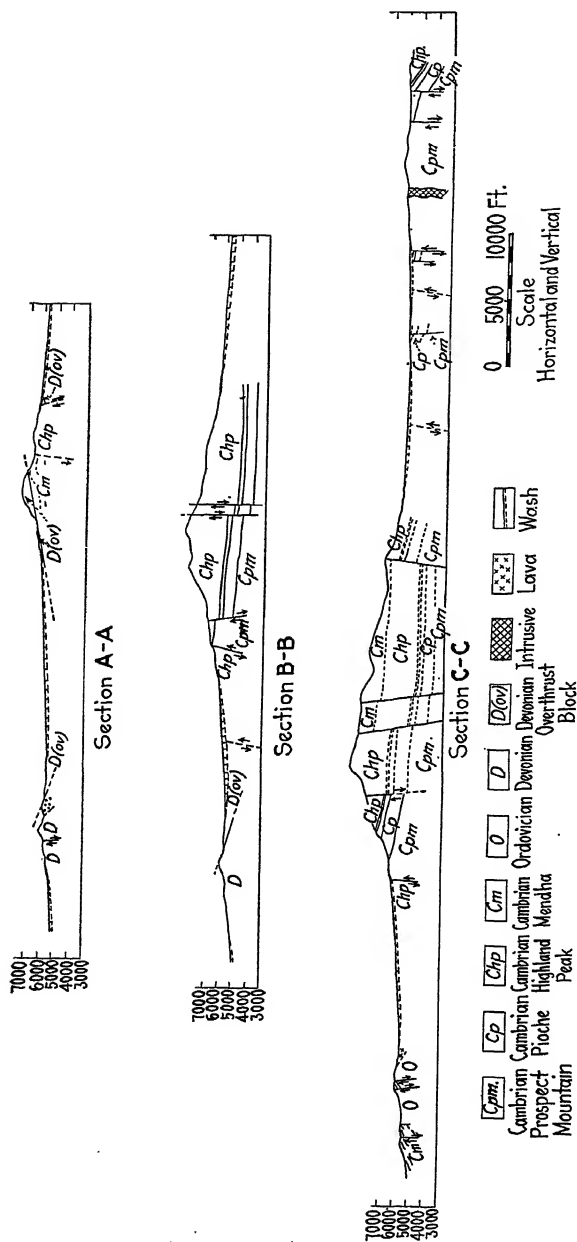


Fig. 3.—STRUCTURE SECTIONS.

Away from the overthrust block it is not possible to tell whether particular faults are earlier or later than the overthrust. Further, any post-overthrust faulting may belong to more than one period.

The general dip of the rocks in the Highland and Panaca quadrangles is to the east, except along the west side of the Ely Springs Range and in the low ranges in the southwest corner of the Highland quadrangle. But the same Cambrian formations appear east to west across the two quadrangles. This can only mean N-S block-faulting of the basin range type with downthrow block to the west. Such faults are believed to separate the Pioche Hills from the Highland Range and the Highland Range from the ranges west. These faults are largely concealed beneath the valley wash, but the faults showing along the west base of the Highland Range belong to this series. They bring the Highland Peak limestone and possibly the Mendha down to the level of the Prospect Mountain quartzite, a drop of certainly 1500 ft. and perhaps two or three times as much. These N-S faults are believed to be responsible for the larger topographic features of the district, though the actual blocks produced by the faulting were reduced to maturity and all fault scarps were destroyed by erosion before the end of Pliocene time. The N-S fault control cannot be followed north of the region about the Bristol mine.

BRECCIATION

Brecciation is common. The overthrust block is so broken that it is generally impossible to get the direction of bedding. The Prospect Mountain quartzite along the Caliente road south of Pioche is faulted, brecciated and recemented by silica to make an even more resistant rock than the bedded quartzite and stands up as rough ledges. The limestones show the same brecciation and recementation by calcite. Often the brecciation accompanies faulting. At Silverhorn, 4 miles northwest of Bristol Pass, brecciated limestones have been replaced by silica and conspicuous ledges made which might be mistaken for quartz veins. The very general brecciation suggests that at the time of faulting the region was under slight load of overlying rocks.

HISTORICAL SUMMARY

The general order of events sketched in the preceding pages may be summarized as follows:

1. Sedimentation during most of the Paleozoic from Cambrian to Pennsylvanian.
2. Uplift, slight warping (the lavas in Condor Canyon have an average dip of 23° , the adjacent Highland Peak limestone of 40° , and this lesser dip of the lavas seems general), and erosion.
3. Volcanism of mid-Tertiary (Miocene) time, giving lavas and tuffs.

4. Tilting and normal faulting.
5. Thrust faulting.
6. Normal block-faulting of the basin range type.
7. Erosion of the faulted blocks to maturity and to essentially the topography of today.
8. Volcanic ash outbursts of probably late Pliocene time and the deposition of several hundred feet of waterlaid tuffs in the valleys.
9. In Meadow Valley: valley cutting, which has produced a bad-land topography in the Panaca region and canyons where the streams cut into the hard rocks.

PART II.—ECONOMIC GEOLOGY*

SKETCH OF THE HISTORY OF MINING

MINING began actively in the Pioche district in 1869. In the next few years phenomenal results were attained and Pioche exceeded in output

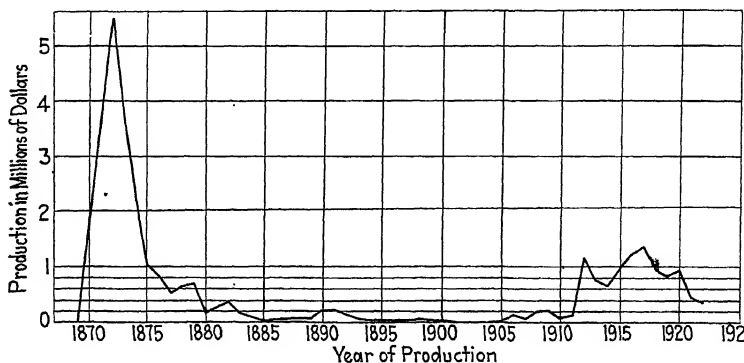


FIG. 4.—PRODUCTION CHART OF THE PIOCHE DISTRICT, NEVADA.

Figures for the production for the years preceding 1905 are from the County Recorder's office, Pioche; for the years from 1905 on they are taken from Mineral Resources of the United States, U. S. Geol. Survey.

all other districts in the West, except the Comstock lode. A town of 10,000 people sprang up in the desert. The history of those stirring years is recounted in lively fashion and with unusual fullness of technical detail by Raymond in his annual reports as U. S. Commissioner of Mining Statistics. The bulk of the output was made by two companies—the Meadow Valley and the Raymond & Ely—which were working on the same vein. The ore was rich in silver chloride and averaged between \$100 and \$200 a ton, or between 75 to 150 oz. of silver to the ton. By 1874, this bonanza vein was largely worked out down to water level, which was

* By Adolph Knopf.

struck at a depth of 1200 ft. An unexpectedly heavy flow of water was found and the ore became zincky. These untoward events, coupled with the exhaustion of the ore above water level, soon after caused the district to go into eclipse. In the 70's and 80's the mineral deposits in the outlying districts were also discovered, notably those in the Bristol district in the northern part of the Bristol Range.

Between the early burst of production and the renewal of activity a long time intervened. In 1907, a railroad was built from Caliente, on the Los Angeles & Salt Lake R. R., and the mining industry began to revive. Tangible results of this revival are shown in the production chart of Fig. 4. Branch lines from the rail terminus at Pioche to the Prince mine and to the Jackrabbit mine allowed these mines to ship to the smelters in Utah large quantities of manganiferous flux low in lead and silver. For a time the Prince mine was the premier producer of lead in Nevada.

In recent years great interest has been aroused by the discovery of large bedded deposits of silver-bearing lead-zinc ore. A few years ago they were without value, but metallurgical research has made them amenable to selective flotation, and has made them the most valuable asset of the district, on which its future appears chiefly to depend.

GENERAL FEATURES OF THE ORE DEPOSITS

The ores are chiefly silver-lead ores where oxidized, and silver-lead-zinc ores where unoxidized. Manganese is a common constituent; in fact some ores are essentially manganese-iron fluxes, and at one mine the ore is high enough in manganese and is so low in lead and zinc that it can be utilized as a manganese ore in iron smelting. Copper and tungsten occur in a few veins.

The ore deposits fall into three fairly distinct groups. First, in point of historic interest and total output, are the fissure veins in the Lower Cambrian quartzite, the veins on which hinged the prosperity of Pioche during the boom days of the early 70's. The fissure veins occur mainly at the town of Pioche, and as they are situated near the Yuba dike of granite porphyry, this association of ore-bearing fissures and porphyry has strongly colored local ideas on the origin of the ores and has influenced development campaigns. These ores are highly oxidized and consist of silver chloride, argentite, cerussite, and galena in a gangue of quartz, chiefly brecciated quartzite. They are comparatively low in lead.

A second group of orebodies consists of lenses in the highly sericitized granite porphyry of the Yuba dike. The metalliferous minerals are like those in the veins in the quartzite but the gangue is soft sericitized porphyry.

A third group is represented by replacement deposits in limestone. This group can be subdivided into replacement fissure veins and bedded replacement deposits.

The replacement fissure veins are steeply dipping and cut across the bedding of the enclosing limestones. They are thoroughly oxidized, for on none of them have the workings penetrated to water level, and are highly manganiferous and limonitic and low in silica. Their size in many places was obviously determined by the intensity of brecciation of the limestone which the fissure traverses. At certain horizons bedded replacement deposits extend out as lateral branches from the fissure veins. Deposits of this type occur mainly in the Lyndon limestone, Highland Peak limestone, and Mendha limestone—a stratigraphic range of 5500 feet.

The bedded replacement deposits that are attracting most attention are replacements of the limestones that are inclosed in the Pioche shale. Some of the most important of these, such as the Combined Metals ore bed, do not outcrop, and were discovered accidentally during mining; and after the principle was grasped that there occur in the district orebodies of a radically different kind than those on which the early prosperity of Pioche was based, other bedded replacement deposits were discovered by systematic diamond drilling.

The geologic position of the bedded replacement deposits, or ore beds as they are locally called, was determined by the intersections of certain steeply dipping fissures with the favorable limestone beds. These fissures themselves may carry some high-grade ore but are commonly poorly mineralized and thin; consequently they are of inconspicuous or insignificant appearance.

Bedded deposits may make off from these fissures at the several successive limestone horizons. At the Prince mine there are at least six such horizons one above the other: four in the Pioche shale, one in the overlying Lyndon limestone, and another still higher, at the base of the Highland Peak limestone.

The ores of the region show no particular relation to the great mass of intrusive granitic rocks, consisting of quartz diorite and allied varieties, that occurs north of Stampede Gap. A broad belt of contact metamorphic rock surrounds the mass, but no productive mines have been developed near it. The immense body of andradite garnet rock at the Manhattan mine, with its large gossans of limonite derived from the oxidation of pyrite and andradite, is manifestly a result of the intrusion.

The nearest intrusive granite to Pioche is a small body of white highly sericitized rock northwest of the Ely Valley mine, 2 miles northwest of Pioche.

Dikes of granite porphyry occur but are scarce. The best known dike is that called the Yuba, which has been highly mineralized and rendered ore bearing. The granitic rocks and the granite porphyry dikes were intruded during a single epoch of igneous activity, at the end of which the ore deposits of the region were formed. At a much later date

dikes of glassy dacite and rhyolite were injected, but this epoch of igneous activity was not followed by an aftermath of ore deposition.

The Pioche region is greatly faulted, and the faulting has occurred at several different times. In common with many other mining districts, the master faults, however, are not mineralized. The displacement along the fissures occupied by the veins is generally small: in a few it exceeds some tens of feet and in others is nil, as in the Mendha vein. In some mines postmineral movement has broken the continuity of the orebodies in closely spaced step-faults.

VEINS IN QUARTZITE

The silver-bearing veins in the Lower Cambrian quartzite were the basis of meteoric results achieved during the early 70's. Raymond could write in 1872 that never in the history of the world had such an output been attained. For a time—a brilliant four years—Pioche was second only to the Comstock lode. The ores averaged between \$100 and \$200 a ton, mainly in silver. From 40 to 85 per cent. of the silver in the ores occurred as chloride. As a rule they were low in lead.

The chief veins that occur in the quartzite are at Pioche and its immediate neighborhood. The only exception is the Comet vein, 9 miles from Pioche, which is interesting and peculiar in that it contains wolframite in noteworthy amount along with galena.

The most productive of the veins in the quartzite was the Raymond & Ely. It was as a rule of moderate width, locally expanding to 9 ft. By 1874 it had already been practically exhausted down to water level at 1200 ft.; subsequently it has been worked to an extreme depth of 1400 ft. It strikes nearly west and dips 75° S, traversing quartzites which dip 20° N. East of shaft No. 3 it split into two branches, known respectively as the Meadow Valley and Burke veins, the angle of divergence between them being 20°.

The filling of the veins in quartzite consists chiefly of angular fragments of quartzite, the metalliferous minerals occupying the interspaces between the fragments. In the upper levels the ore is porous due to leaching, probably in the main the result of the removal of sphalerite and pyrite; it contains lead carbonate, silver chloride, and other oxidation products in minor quantity. In less thoroughly oxidized ore galena appears, and ore from below water level contains black sphalerite.

In recent years the most brilliant result attained by working the quartzite fissure veins was at the Stindt and Donohue lease on the Wide Awake group, 2 miles south of Pioche. In 1919 there was shipped \$120,000 worth of ore, averaging \$60 a ton in silver and lead. The ore, although in a quartzite fissure, is unusual in that it occurred in a vein that dips at angles as low as 20 to 30°. Because of this low inclination, the ore has been erroneously called "bedded ore," but it occupies a fault

fissure, probably a thrust fault, which cuts through the quartzites, here dipping 45° N.

Considerable development work is now being done by the Pioche Mines Co. in cleaning out and extending the old workings on a number of the quartzite fissure veins that are associated with the Yuba dike, such as the Mazeppa, Chapman and others.

OREBODIES IN GRANITE PORPHYRY

Shoots of ore occur in a granite porphyry dike known as the Yuba dike, which cuts through the quartzite on Treasure Hill south of town. The beliefs that have grown up concerning the association of ore and this dike have been warmly cherished in the district. It has often been asserted that the Yuba dike is the source of all the ore in the district and that consequently the orebodies become poorer the farther they are away from the dike. One of the results of this belief is that considerable expensive exploration has been done in the vicinity of other dikes which have been mistakenly regarded as similar to the porphyry of the Yuba dike. For example, certain dikes of glassy andesite, dacite and rhyolite have been thought to be like the granite porphyry of the Yuba dike and hence to be likely as ore-bringers; but as a matter of fact they were intruded long after the time at which the ore of the district was formed. That these dikes are younger than the ore is best shown at the Hamburg mine, where a glassy dacite (technically a hornblende-biotite dacite vitrophyre) cuts through the orebody.

The Yuba dike contains shoots or pods of ore distributed irregularly through it; they occur sporadically at both contacts or anywhere in the interior of the dike. The dike is 40 ft. thick, trends easterly, and dips 80° S. It cuts through the quartzite, which dips 25° N, and has been traced as far east as the Alps mine.

The dike consists of biotite granite porphyry that has been highly sericitized; practically the only original mineral left is the quartz that occurs as many small phenocrysts dotted through the rock. Where unoxidized the porphyry contains finely disseminated pyrite.

In general the dike is shattered, and at intervals it contains thin lenses of ore lying in its plane and bounded by well-defined walls. The quartzite along the contact of the dike is shattered, and in places "quartzite fissures" extend out into the quartzite from the dike at an angle of 45° with its course.

Genetically, it is clear that the Yuba dike was shattered after it had consolidated, and that it then served as a channel way along which hot solutions ascended. These solutions deposited galena, sphalerite, pyrite and some silver-bearing mineral and altered all the feldspar and biotite to white silky mica (sericite).

A large amount of work was done on the Yuba dike and its related quartzite fissures by the "old timers" and subsequently by lessees. Most of it was done on small tracts, on many of which deep shafts were sunk, some as deep as 1200 ft. As a result the ore that was taken out could have been found with one-third of the work that was actually done. At the present time most of the ground is owned by the Amalgamated Pioche Co. and is under lease to the Pioche Mines Co.

REPLACEMENT DEPOSITS IN LIMESTONE

The replacement deposits in limestone comprise two kinds: (1) Replacements along fissures that cut across the bedding of the limestone forming replacement fissure veins; and (2) replacements of certain limestone beds. Replacements of this kind are consequently tabular ore-bodies that lie parallel to the bedding, the so-called bedded orebodies. In some mines both kinds are associated together in one and the same ore system, for at certain horizons tabular masses of ore extend out from the ore-bearing fissure into the limestone, like branches from the main trunk of a tree.

The ore deposits are silver-bearing galena-sphalerite-pyrite bodies. They range from solid sulfide ores to ores that are more or less manganiferous. The manganiferous ores are highly oxidized, consisting of pulverulent oxides of iron and manganese and containing a negligible amount of lead, zinc and silver. In fact, pyrolusite (MnO_2) is sufficiently abundant at the Jack Rabbit mine and lead, zinc and silver are so nearly absent that the material is shipped to the iron smelter in Utah near Provo, as manganese ore.

The most notable representatives of the replacement fissure veins are in the Bristol district. The May Day, Gypsy, Tempest and Hillside veins are among the most prominent. They yield silver-bearing copper-lead-zinc ores, all highly oxidized. Galena in small amount is the only sulfide found. Contrary to an opinion expressed that these ores were deposited from downward-moving solutions,⁴ there is no reason to doubt that the primary ores were deposited from ascending solutions. During oxidation, however, there has been some rearrangement of the metals, but not much, on account of the prevalence of limestone. The copper has tended to move toward the footwall, where it was precipitated as chrysocolla and copper pitch; iron, manganese and zinc have also probably moved a little, but the lead, altered from galena to cerussite, has remained in place.

⁴ J. M. Hill: Notes on Some Mining Districts in Eastern Nevada. U. S. Geol. Surv. Bull. No. 648 (1916) 129.

BEDDED ORES

In recent years the bedded ore of the Combined Metals mine at Pioche has been of main interest. During 1926 this mine was producing at the rate of 4000 tons a month. The ore is practically a solid sulfide aggregate that consists of an intimate intergrowth of pyrite, sphalerite and galena. The pyrite predominates and the zinc content is 14 per cent., lead is 7 per cent., silver is 7 oz. to the ton, and gold is only 0.03 ounce to the ton. This ore was worthless until a proper flotation was devised. So intimately are the sulfides intergrown that the ore must be ground to 200-mesh in order that the lead and zinc can be separately concentrated by selective flotation.

The orebody occurs at the intersection of a steep fissure (the Greenwood) and a limestone bed dipping 20° that occurs in the Pioche shale. It extends out laterally on both sides of the fissure as much as 100 ft. The ore has clearly been formed by replacement, as all the peculiarities of the original rock, such as its distinctive nodular character, are retained and preserved. The orebody is a tabular mass 30 to 40 ft. thick, thickest adjacent to the Greenwood fissure and tapering down away from it. The downward extension of the orebody is determined by the angle of intersection that the Greenwood fissure, which strikes N 65° E and dips 65° N, makes with the flat southwesterly dipping limestone bed.

The Greenwood fissure has yielded some high-grade ore near the replacement ore bed. It is now known as a result of recent diamond drilling that the Greenwood fissure has caused replacement by ore in another limestone bed lying 118 ft. above the main ore bed, but the amount of this ore is not yet known. It is an interesting fact that the Greenwood fissure and the Raymond & Ely fissure are adjacent, trend nearly parallel, and dip in opposite directions. Whether the Raymond & Ely vein has caused the development of replacement ore where it intersects the limestone beds in the Pioche shale is not known.

The bedded replacement ores at the Prince mine, 3 miles southwest of the Combined Metals mine, differ considerably in composition and degree of oxidation from that at the Combined Metals mine. Those above water level are thoroughly oxidized and consist of manganese-iron oxides low in silver, lead and zinc. About 800,000 tons of ore carrying $2\frac{1}{2}$ to 3 oz. of silver per ton, 3 per cent. lead, 35 per cent. iron, and 15 per cent. manganese has been shipped. Ore of this kind contains also 1 per cent. of zinc, which vitiates its possible use as a manganiferous iron ore. The bed known as the 20-ft. bed, which is in the Pioche shale near the top of that formation, has been the most profitable. It was mined for a length of 1500 ft. and over a maximum width of 400 ft. The bed was mined on the contract system and the ore was delivered at

the shaft for 40 c. a ton. Besides the 20-ft. bed four other beds of manganiferous flux were worked.

Unoxidized sulfide ore was found by diamond drilling to occur as a replacement of one of the lower limestones in the Pioche shales. This bedded ore is 340 ft. below water level, which stands at 496 ft. below the collar of the Prince shaft. The ore consists of sphalerite, galena and pyrite in a gangue of manganiferous siderite and minor quartz. Oxidation of such a manganiferous siderite and pyrite would account for the abundant iron and manganese in the bedded ores that are above water level. Another sulfide bed 245 ft. below this one is indicated and appears to be only 10 ft. above the Prospect Mountain quartzite.

The mineralizing fissure from which these seven successive horizons of bedded ore were developed appears to be the Great Western fault. Along this fault considerable vertical movement took place, part of which was postmineral; it did not, however, exceed 400 ft., for Lyndon limestone, which is only 400 ft. thick, occurs at the surface on both sides of the fault.⁵

CONCLUSION

The bonanza output of the Pioche district came from the fissure veins in the Lower Cambrian quartzite. In 4 years the two mines that worked the principal vein—the Raymond & Ely vein—were able to disburse \$4,981,000 in dividends, despite the great handicaps placed on them by what was then a remote desert mining camp. The known ore was rapidly exhausted, but it was mined skilfully in spite of its richness, and the ground was carefully explored for more ore.⁶ The upper levels were gone over several times. Although the grade of the ore then mined was much higher than would now be necessary for profitable working, yet in view of the explicit statements in Raymond's reports and by analogy with other Nevada camps which have been found to have had most or all of their ore carefully mined out by the "old-timers," it seems overoptimistic to expect to find bodies of bonanza ore or extensive quantities of moderate grade ore in the old-time workings.

There is a possibility that some of the fissures that are thin and feebly mineralized in the Pioche shale may become stronger and carry ore where they extend down into the underlying quartzite formation. The bonanza ore of the Raymond & Ely vein did not come to the surface at the Lightner shaft, but was cut at a depth of 200 ft., after the shale which there overlies the quartzite had been penetrated; the ore shoot the top of which was thus cut yielded several millions. And at other localities the distinctly adverse effect of the Pioche shale on the veins is clearly shown.

⁵ Sections used at the mine show a supposed vertical displacement of 800 ft.

⁶ R. W. Raymond: Mineral Resources West of the Rocky Mountains for 1872. (1873) 180.

The recognition in recent years that large replacement orebodies have formed in the limestones interbedded with the Pioche shale where the limestones are cut by certain mineralized fissures has greatly enhanced the possibilities of the district. The limestone (the "Combined Metals ore bed") 250 ft. above the base of the shale has so far been found to be the most productive horizon. As demonstrated at the Prince mine, one and the same fissure can have caused replacement orebodies to form at five or six successively higher limestone horizons.

The successive limestone beds thus replaced occur not only in the Pioche shale but also in the overlying Lyndon limestone and in the still higher Highland Peak limestone. Accordingly, it may be worth while to explore to considerable depths those mineralized fissures that yield ore in the formations above the Pioche shale, on the chance that the lower favorable limestone horizons, especially those in the Pioche shale, have been replaced by ore.

The practicability of prospecting these lower limestone beds depends, among other factors, on the depth at which they occur below the surface or below the deepest workings of the particular mine in which such prospecting is contemplated.

The geologic column, showing the order of superposition of the various formations and their thicknesses, is therefore of supreme practical importance. On account of the acutely faulted condition of the region, especially in the immediate vicinity of Pioche, the construction of this column became possible only after extended areal surveys of the whole region by Professor Westgate, and it is one of the chief objects of this preliminary paper to make this column available for such exploratory work. On it may be seen that the Combined Metals bed, so far the most favorable bed known, is 870 ft. below the top of the Pioche shale and about 400 ft. below the bottom of the Highland Peak formation.

That large valuable bodies of bedded replacement ore have formed adjacent to insignificant fissures has already been pointed out: it is beyond doubt one of the impressive features in the geology of the district. It opens the possibility that there may occur many such bedded deposits that do not crop out. The discovery of these orebodies will be difficult, but it will be facilitated primarily by applying skilfully a knowledge of the geologic column and by determining the faulting that has altered and disturbed the normal sequence of the strata.

Subsidence and Its Relation to Drainage in the Red Iron Mines of the Birmingham District, Alabama*

By W. R. CRANE,† BIRMINGHAM, ALA.

(New York Meetings, February, 1925, and February, 1927)

THE effect of mining in the red-ore mines of the Birmingham district has been observed for some time, but, except in a few localities, little difficulty has been experienced from disturbance of cover. Cave-ins near the outcrop and fracturing of the surface at greater distances are the most pronounced manifestations of disturbance, while limited areas at points distant from the outcrop and under greater depth of cover have actually subsided.

Were it not that the orebed is overlain by one or more water-bearing formations, the fracturing or settlement of the surface would not be serious; but because of these formations, fracturing of the top rock may in itself be of sufficient importance to warrant the adoption of protective measures. Further, the collapse of pillars over a wide area may develop squeezes that, if not controlled, may jeopardize the integrity of the mines and necessitate a change in development, from slopes to vertical shafts, at a much earlier period than is now contemplated.

Drainage in these mines has been discussed in papers on mining practice,¹ but the fact that water, varying from 0.16 to 3.46 and averaging 1.43 times the amount of ore mined, has to be pumped from the mines daily, indicates the importance of the problem and the advisability of making a thorough investigation of the source of mine water, its mode of entry into the mines, its effect upon mining conditions, and consideration of means of improving the conditions.

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† Mining engineer, U. S. Bureau of Mines.

¹ W. R. Crane: Iron-Ore (Hematite) Mining Practice in the Birmingham District, Alabama. *Bull.* 239, Bur. Mines (1926).

W. R. Crane: Red Iron Ore Mining Methods in the Birmingham District. *Trans.* (1925) 72, 157.

W. R. Crane: Roof Support in the Red Iron Ore Mines of the Birmingham District. *Trans.* (1925) 72, 187.

Much detailed information has been collected regarding: rainfall, and run-off in streams draining the district; the effect of different kinds of formations, porous and non-porous; the inclination of the surface; the condition of the surface above the mines, i. e., whether or not disturbed by ground movement; the occurrence of the ore and character of associated formations, particularly with respect to presence of jointing or slip planes, faults, etc.; failure of top rock in the old stopes; and the conditions under which water effects entrance into the mines, i. e., whether as a natural or forced flow.

ACKNOWLEDGMENTS

Data and suggestions have been received from many persons, who have been or are now actively engaged in mining operations in the district and particularly from the following: Members of the advisory committee to the Bureau of Mines; W. J. Penhallegon, general superintendent of ore mines and quarries of the Republic Iron & Steel Co., and chairman of the advisory committee; C. E. Abbott, general manager of mines of the Tennessee Coal, Iron & Railroad Co.; C. E. Bowron, chief engineer of the Gulf States Steel Co.; W. C. Chase, general superintendent of the Woodward Iron Co.; J. E. Strong, vice-president of the Alabama Co.; and W. J. Thomas, general superintendent of the Sloss-Sheffield Iron & Steel Co. Local superintendents and engineers of the various companies, also, have freely given their assistance.

GENERAL CONSIDERATIONS

In the Birmingham district, the area affected by fracturing and settlement of the surface through collapse of mine workings is on the southern and eastern slope of Red Mountain in Shades Valley and, aside from an area 4 or 5 sq. mi. in extent, comprises no land of particular value for building or farming; further, because of the increase in thickness of cover due to dip of orebed, it is not likely that serious disturbances will result from settlement because of mine workings at any considerable distance beyond the slope of Red Mountain.

FACTORS CONTROLLING MINE SUBSIDENCE

The breaking of the top rock forming the roof of the workings and the failure of pillars, with the resultant collapse of overlying formations, are due directly, of course, to the removal of portions of the orebed; but other factors, affecting the direction and to some degree the extent of the failure of the overlying formations, are closely related to the occurrence of the ore. Of the factors involved, aside from mine openings that affect the collapse of mine workings and the fracturing and settlement of the surface, slip planes exert the most influence and often control both

the vertical and the horizontal direction of the lines of failure. Other less known factors affecting subsidence are faulting and warping of the orebed and associated formations, which effects are not directly evident but are manifest in the prominence and position of slip planes. Jointing or slip planes then represent the accumulated action of a wide range of largely unknown factors and, being susceptible of observation and measurement, are the most important elements to be considered in the study of controlling factors in mine subsidence.

With two prominent slip planes, the orebed is broken into blocks varying from 1 ft. to 8 and 10 ft. across, the average being close to 3 or 4 ft. When a number of sets of slip planes occur, the orebed is broken into a large number of blocks, varying both in size and shape, according to the direction taken by the respective slip planes.

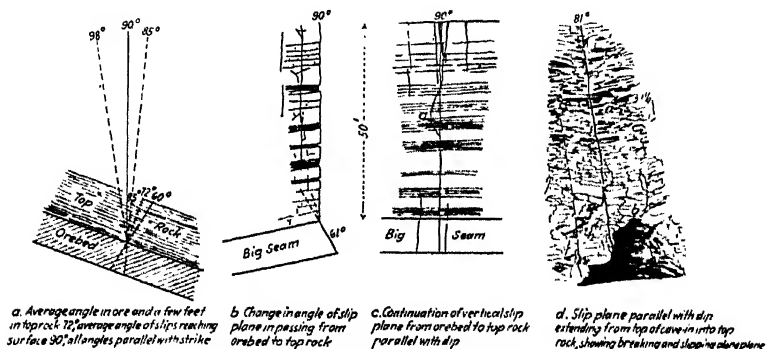


FIG. 1.—DIRECTION TAKEN BY SLIP PLANES IN TOP ROCK.

Observations on bearings of slip planes on the surface show a relatively wide range in direction taken, but under normal conditions there is a remarkable uniformity. Similarly, with respect to the vertical position or dip of slip planes; while quite a range has been noted the greater number observed and measured show a surprising uniformity to certain angles, with limits close to the mean or vertical; see Fig. 1.

The dip of surface cracks was obtained by probing with a steel bar, the inclination being taken from the bar. When making these observations, it soon became apparent that the angles obtained represented the position of slip planes, the well-defined break of the slip planes showing, in many instances, at the surface, except where the surface was covered with debris, or, as often occurred, several feet of Fort Payne chert remained in place. It is interesting to note that the angle of dip of the slip planes is with the dip of the orebed at both ends of the field and is opposed to it in the middle of the field.

Owing to the forces that produced the slip planes, or as a possible after-effect, the angles assumed by them in the orebed are considerably

less than in the top rock, ranging from a minimum of 60° with the horizontal, in the former, to the vertical, in the latter. The majority of jointing planes or slips are within limits of 10° from the vertical.² Two illustrations are given of the vertical position assumed by slip planes in the top rock; in *b* (Fig. 1), the change in angle is noted in a plane parallel with the strike; in *c* and *d*, the slip planes are parallel with the dip. The angles given in *a* are the average of many hundreds of observations on slip planes parallel with the strike; a similar relation may obtain with the slip

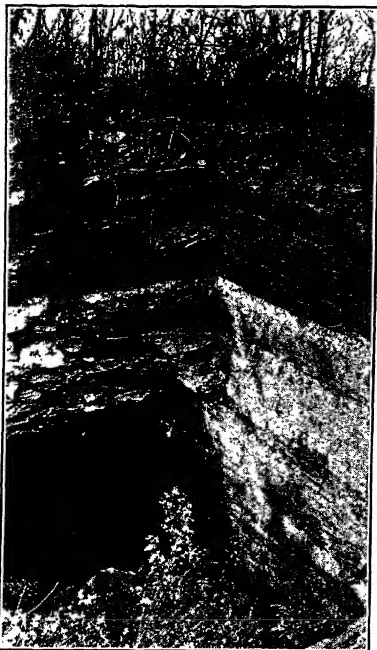


FIG. 2.—BREAK OF TOP ROCK ALONG SLIP PLANE.

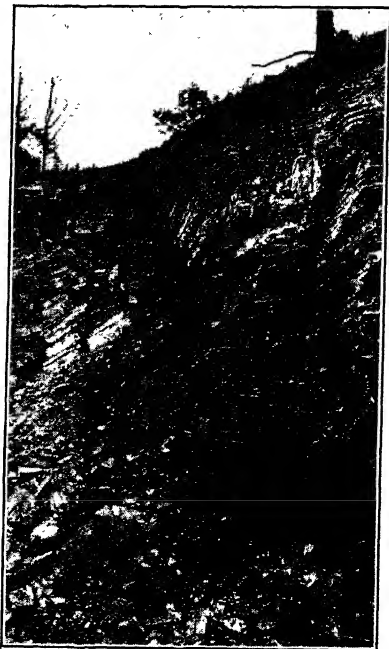


FIG. 3.—CAVING WALL OF OPEN CUT FOLLOWING SLIP PLANES.

planes roughly paralleling the dip, but because of the relatively limited number of occurrences available for observation, the facts have not been established.

In order that the relation between the slip planes at the surface might be compared with those in the orebed and adjacent formations, other observations were made in the open-cut workings and underground.

Whatever action caused slip planes is immaterial in this connection, but the effect is profound; they cut not only the 20-ft. orebed but extend

² Oliver Bowles: The Technology of Marble Quarrying, Bur. of Mines, *Bull.* No. 106 (1916) 27.

vertically through the overlying and underlying associated formations for several hundred feet above and below. See Figs. 2 and 3.

CAUSE OF MINE SUBSIDENCE

The primary cause of fracturing and settlement of formations overlying mine workings is the failure of roof and roof supports, or pillars, in the openings made for the extraction of ore. The openings may be unduly large, causing failure of top rock in a relatively short time; or moderate sized openings may result in failure after a long period of time. In either case the result is the same so far as effect on the surface is concerned; but surface disturbances resulting from the failure of top rock may be markedly different in the two cases, *i. e.*, sudden collapse through failure of adequate support lacks the regularity and symmetry that might be expected and is observed in the slow disintegration and failure over extended periods of time. To determine the order, if any exists, in the failure of top rock in mine workings, a series of observations were made, covering a period of a year or more and embracing practically all the active mines of the district. As the failure of top rock precedes the fracturing and settlement of the surface, it is desirable to consider underground conditions prior to the effect of such failure on overlying formations and the surface.

FAILURE OF TOP ROCK IN THE MINE WORKINGS

The sudden collapse of roofs of stopes immediately following the removal of ore may occur but it is unusual, to say the least, and is the result of either abnormal conditions of occurrence of top rock, as the presence of prominent slip and crossbedding planes, or breaking the top rock by the use of too much powder; it is seldom the result of excessive width of workings.

The failure of top rock, as well as pillars of ore left as supports, acts slowly, as a rule, months and years often being required for the ultimate collapse. The cause of the failure of both roof and pillars is the weight of the overlying formations acting directly on the top rock forming the roof of the workings and the supporting pillars, through drag in the first instance and heave in the second, bringing about their disintegration and failure.

The cause of failure of top rock having been determined,³ attention was directed to the order of occurrence so that definite facts might be secured. Angles of break of top rock were measured, several hundred

³ W. R. Crane: Roof Support in the Red Ore Mines of the Birmingham District, Alabama. *Trans.* (1925) 72, 187.

observations having been made and averages for the localities plotted as shown. In Fig. 4, graphs A_1 , A_2 , B_1 , B_2 , C_1 , and C_2 show the angles of break at the top and bottom of stopes or at the upper and lower ribs; *i. e.*, on the dip and in the headings, or on the strike of the orebed. The average lines are shown in Fig. 5. In Fig. 4, the individual plottings are given, which are omitted from Fig. 5 for clearness. Lines A_1 and B_1 give the initial angles of break of top rock at the top and bottom of stopes and show no marked deviation from straight lines as d and e , while the final angles of break at similar points give broken lines, represented by the lines A_2 and B_2 , the average lines being c and b . The percentage

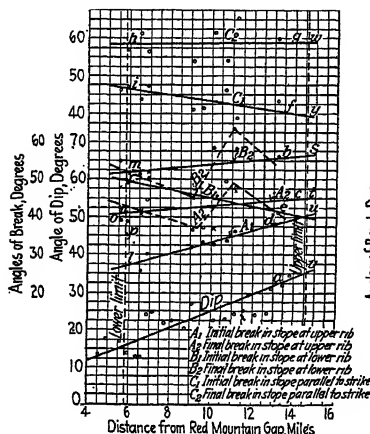


FIG. 4.—ANGLES OF BREAK IN CAVING STOPES.

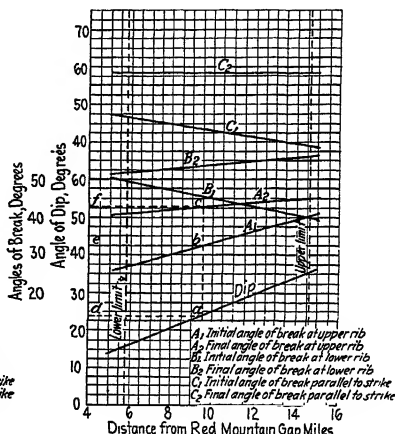


FIG. 5.—DETERMINING ANGLES OF CAVING IN STOPES.

of slate in the top rock corresponds remarkably close with the final angles of break, as given by the angles observed in the caved stopes; this would seem to indicate that slate, being more friable than sandstone, has a pronounced effect on the angles of break in caving top rock. The initial angle of break in top rock C_1 (Fig. 5) varies somewhat with the dip of the orebed, while the final angle of break C_2 is not affected by the dip of orebed; the average lines are f and g (Fig. 4).

The initial angle of break in top rock parallel with the strike C_1 and the initial angle of break at the bottom of stopes B_1 , are similar, decreasing with increased dip of orebed. Aside from the two exceptions mentioned, the angles of break of top rock increase with increased angle of dip of orebed. See Fig. 6.

The reason that the angles of break of top rock at the bottom of stopes, as represented by lines e and b , do not conform more closely, like d and c at the top of stopes, is that the caved material in stopes banks against the

roof, particularly on steep dips, and prevents the breaking down of the top rock. Subsequent clearing of stopes by failure of pillars or escape of caved rock through upsets permits the final angle of break to be formed.

Having determined the angles of break in top rock, it is possible to obtain, from (Fig. 5), important data relative to conditions existing in

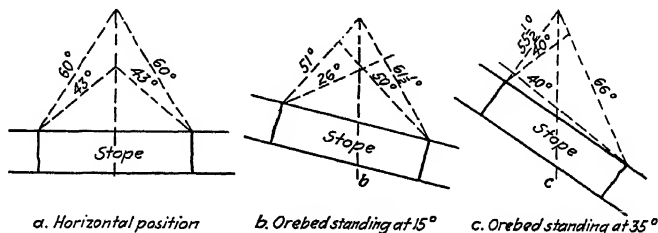


FIG. 6.—INITIAL AND FINAL ANGLES OF BREAK AT VARIOUS ANGLES.

various parts of the district, the relation between angles of break and dip of orebed also being shown. For instance, $9\frac{1}{2}$ miles south of Red Mountain Gap, the dip of the orebed is 24° and the initial and final angles of break of top rock at the top and bottom of the stopes are $32\frac{1}{2}^\circ$, $45\frac{1}{2}^\circ$, 53° , $63\frac{1}{2}^\circ$, etc.

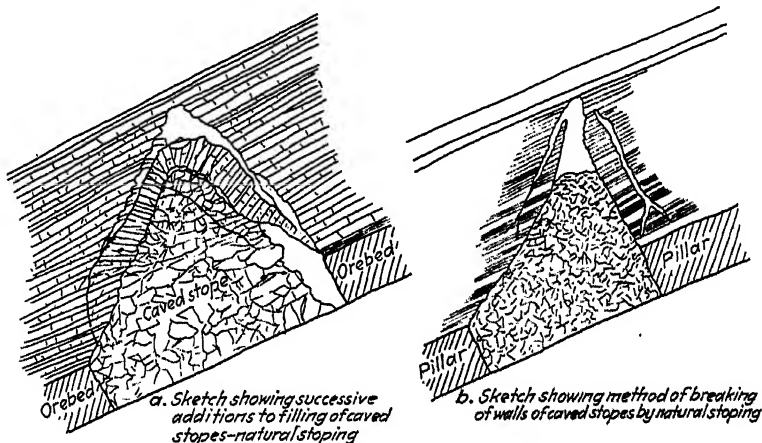


FIG. 7.—FAILURE OF TOP ROCK BY NATURAL STOPING.

The term "final break" refers to the second stage of the break of top rock, which might be more appropriately defined as an intermediate angle of break between that assumed in extensively caved stopes and the final collapse of top rock, resulting in subsidence, when the break may become vertical or practically so, which is particularly true in this district

where ultimate failure of top rock is largely controlled by slip planes (Fig. 1).

The direct result of the increase in width of caved stopes by natural stoping is an increase in height of the caved area, especially by the merging together of two or more stopes. See Figs. 7 and 8. The increase of height of a single stope under light cover usually results in a cave-in, forming a round or elliptical opening to the surface; or, in some instances, several hundred feet of stope may cave to the surface, forming a long narrow break. But when several adjacent stopes are merged, in direct line up or down the dip of the orebed, a serious disturbance of top formations results; the

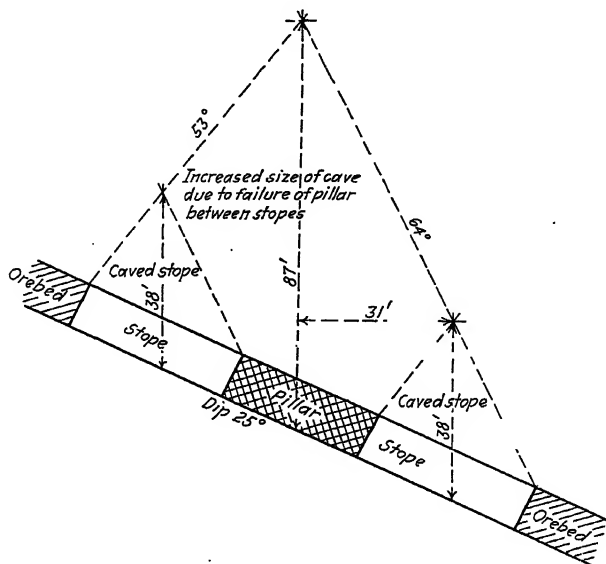


FIG. 8.—MERGING OF STOPES RESULTING IN SUBSIDENCE OF SURFACE.

caved ground extends for a considerable distance vertically and may reach the surface even though the cover is several hundred feet thick (Fig. 8). Under such conditions, fracturing of the surface is the direct evidence of extensive caving underground, which is usually accompanied by more or less pronounced settlement or subsidence of the surface. Fig. 8 represents conditions as they exist today where fracturing and settlement occur under slight cover, but have also been observed under much greater depth of cover, as when the final angles of break in caving stopes intersect at considerable distance below the surface.

The thickness of the orebed worked contributes largely to the disintegration of the top rock and its subsidence, the thicker the bed the greater the amount of movement.

The next important consideration, in connection with caving stopes, is in relation to the order or system in caving; *i. e.*, what are the factors, aside from the position of the stope, influencing or controlling the direction taken by the caving ground? Observations in caved stopes, which often extended into the overlying rocks for a hundred feet or more, show that, to the limits of possible observations, slip planes exert a dominant influence on character and extent of breaks both horizontally and vertically.

DIRECTION TAKEN BY BREAKS

The failure of top rock along horizontal lines, as influenced by and in fact controlled to a large extent by slip planes, is well known. The longer breaks along slip planes are approximately parallel with the strike or headings, while the slips making almost right angles with them run with the dip. The angle of break in the top rock does not follow the

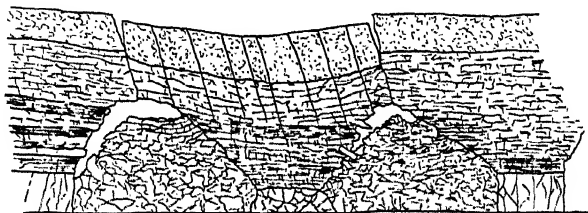


FIG. 9.—FAILURE OF TOP ROCK ALONG SLIP PLANES WITH RESULTANT SETTLEMENT OF SURFACE.

line of the stope but forms an approximate mean between the strike, or headings, and the slip planes roughly paralleling the dip. The breaks in the sandstone are pronounced, while those in the more friable slates are less distinct; in fact they form almost straight lines, practically bisecting the faces of the breaks along slip planes in the sandstone above.

As the effect of slip planes in controlling the horizontal breaks of top rock is second only to that of the lines of pillars limiting the size and shape of stopes, so they influence the direction taken by breaks extending more or less vertically into the top rock. That slip planes parallel with the strike assume the vertical position in the top rock has been shown (Fig. 1), but it has not been established that the slip planes paralleling the dip assume the vertical in the top rock, although that they approach thereto is well known. The failure of top formations would naturally be expected to follow lines of weakness and actual breaks in the rocks, as furnished by slip planes, which is actually the case; see Figs. 2 and 3. Again, observations at the tops of extensive caves in stopes show the prominence and persistency of slip planes throughout the overlying formations and their influence on the weakening and failure of the rock layers.

It is evident, then, that lines of break in the top rock develop at the top of caved stopes following slip planes in vertical and horizontal lines through the formations. Caves in stopes ranging from 50 to 100 ft. in height materially weaken the overlying ground and cause it to break readily at the weaker points; see Fig. 9.

FRACTURING AND SETTLEMENT OF SURFACE

Having established the fact that a relation exists between slip planes and breaks on the surface, as shown by cracks, many observations were made on the bearing of both cracks and slip planes, which are shown in

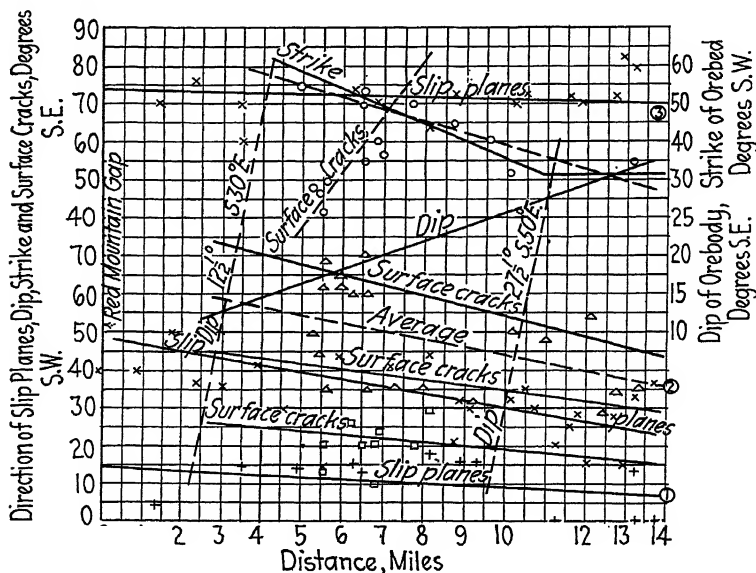


FIG. 10.—BEARINGS OF SLIP PLANES AND SURFACE CRACKS.

Fig. 10. The data recorded represent scores of observations for each locality plotted and are not single observations, as might appear.

At the bottom of the diagram are lines for average angles of bearing of S. 9° and 31° W. for slip planes; also average angles of bearing of surface cracks, corresponding to the above bearings of slip planes, being S. 19° and 45° W. The bearing S. 45° W. is an average between two well-defined average lines corresponding to the angle of bearing of S. 31° W. for the slip planes. At the top of the diagram is given the average angle of bearing of slip planes of S. 72° E., with the corresponding bearing of S. 54° E. for surface cracks.

The diagram shows the bearings of strike, dip, slip planes, and cracks on the surface, as observed and measured at various points along Red

Mountain; also the relation of cracks of different bearing to each other, to slip planes, and dip and strike of orebed. The parts of the field affected by fractures and settlement are also indicated.

While it has been shown that the failure of top rock is along slip planes, the shape and direction of the workings contribute largely in determining the direction taken by breaks and cracks on the surface. Where the direction of one of a set of slip planes coincides with that of headings and stopes, the most favorable condition obtains for the formation of breaks in the top rock. A less favorable condition is where the direction of both of the set of slip planes diverges considerably from that of the workings, although the direction of break in the top rock may still conform closely to that of the workings. However, the second of the set of major slip planes, in either case mentioned, exerts a pronounced influence on the direction taken by the break in the top rock and, to all

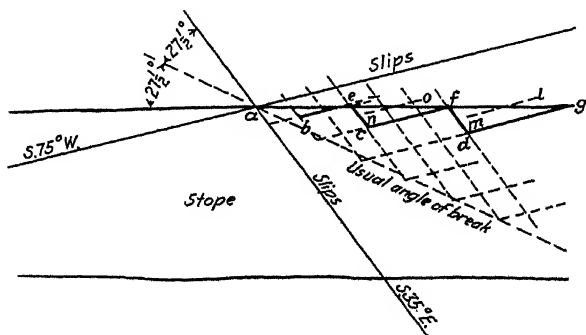


FIG. 11.—SKETCH SHOWING HOW TOP ROCK FAILS ALONG SLIP PLANES HORIZONTALLY AS A MEAN OF LINE OF WORKINGS AND ANGLE OF SLIP PLANES.

intents and purpose, the result is the same as though only one of a set of slip planes cut across the strike or the line of workings, the other of the set paralleling the workings.

With both of the set of major slip planes of equal or approximately equal prominence cutting both orebed and top rock, the break resulting from failure of top rock in the workings and formation of cracks on the surface is roughly a mean between the slip planes; or to be more exact, it is a bisector of the angle formed by the crossing of the line of workings or strike by the second slip plane; see Fig. 11.

MODE OF FAILURE OF TOP ROCK

The thickness of cover is a controlling factor in the character and extent of surface disturbances. With light cover, the failure of the top rock in a stope may reach to the surface; while with considerable depth of cover fracturing and settlement of the surface may result. The

depth of cover at certain distances from the outcrop of the Big Seam and above it on Red Mountain is shown in Fig. 12. The present

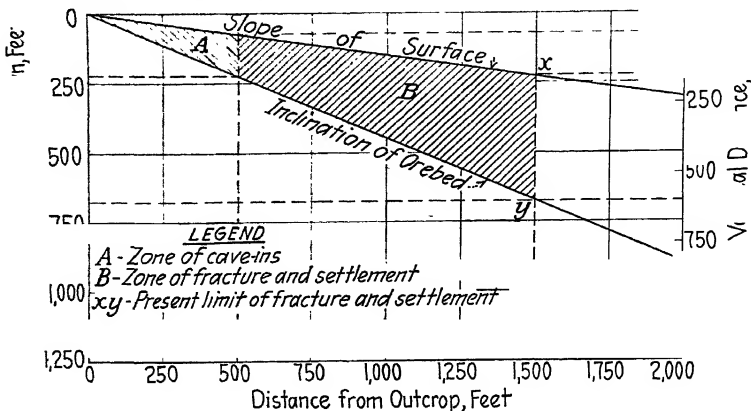


FIG. 12.—DEPTH OF COVER AT VARIOUS DISTANCES FROM OUTCROP AND PRESENT ZONES OF DISTURBANCE.

zones of disturbance and fracture are also given, showing that no disturbance has been observed beyond 1500 ft. from the outcrop with a thickness of cover of 400 ft. It is doubtful whether surface breaks will ever occur



FIG. 13.—SURFACE CRACK FOLLOWING SLIP PLANES.

beyond a depth of 750 to 800 ft., although subsidence will take place, particularly if a high percentage extraction of ore is made, with ultimate collapse of top rock.

In Fig. 13 is shown a crack that developed with light cover and followed the slip planes closely, breaking across from one line of planes to another in order to maintain the direction determined by workings or the mean between the workings and the slip planes crossing them at moderately wide angles.

As previously indicated, certain surface breaks or cracks develop at the top of stopes, the breaks following the vertical slip planes to the surface; see Fig. 9. It has been observed along the mountain slope in the direction of dip of the orebed, that cracks following the direction of the workings are spaced roughly at more or less definite intervals, which are the distances between headings.

A break at a certain point may stand open for weeks or months before another break results from failure of pillars and top rock in the upper



FIG. 14.—FAILURE OF GROUND PARALLELING CAVING SLOPE.

workings; but when subsequent breaks occur, the mass of ground lying between them settles back in the direction of the surface slope and dip of orebed and closes the first crack or cracks formed. By successive breaking of top rock along the lines of slip planes and the letting go of large masses of ground, the phenomenon of opening and closing of surface cracks is accounted for.

A most striking example of the failure of top rock with consequent fracturing of the surface in the general direction of the dip is given in Fig. 14. In this case, however, slip planes contributed to the direction taken by the surface cracks. The haulage slope was driven on slip planes, as was the custom of development in the early days, so that to the line of weakness caused by the caving of top rock above the slope was added the effect of slip planes, the combined result being that blocks of ground broke loose and swung toward the slope.

It is interesting to note that many of the cracks observed in the areas affected, parallel or approximately parallel the deep cuts or runs on the slope of the mountain under which mining has been done. The reason for this is obvious, for the runs have cut deep into the formations overlying the orebed, often reaching within a comparatively few feet of the orebed itself. The slopes of the sides of the runs are steep and similar in effect to the slopes on which cracks are developing. The masses of ground on both sides of the runs break along lines of slip planes, lines of failing top rock over stub pillars and in some cases along haulage slopes, and swing in toward the runs developing cracks approximately parallel with them; see Fig. 15.

The step following extreme fracturing of the top formations and breaking of the surface is subsidence; but, unless the supporting pillars

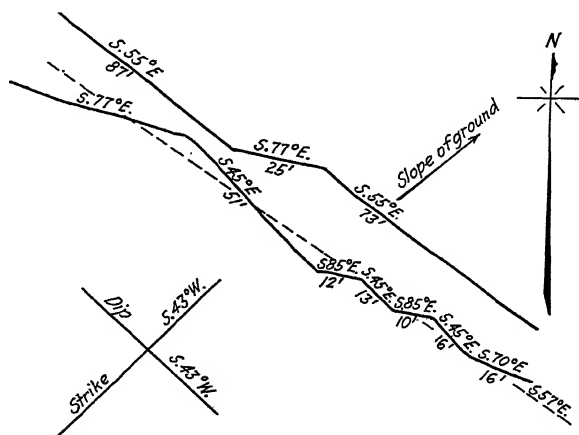


FIG. 15.—EFFECT OF SLOPE ALONG RAVINE ON FORMING SURFACE CRACKS.

have failed on a large scale and over considerable area, no pronounced settlement is observed. Further, the subsidence noted will become less with increased depth of cover, the surface settling as a whole with little or no fracturing of the top formations.

With slight covers, the caving of one stope may result in a cave-in, two or three adjacent stopes caving may produce extensive fracturing of the surface under moderately thick covers, while a general collapse of pillars with consequent disturbance of top formations will produce more or less serious subsidence; see Fig. 9. The first two conditions are of common occurrence, with accompanying cave-ins and surface cracks; the last and final stage, while hardly evident at present, will come with the general and uniform collapse of the old worked-out areas under considerable depth of cover.

EXAMPLES OF FRACTURING AND SETTLEMENT OF SURFACE

The controlling factors responsible for the disturbed condition of the top rock and formations overlying the orebed having been given in detail, corroborative examples may now be set forth. A number of typical areas have been mapped, showing the relation of slope of surface, effect of slip planes, and a combination of both in forming breaks at the

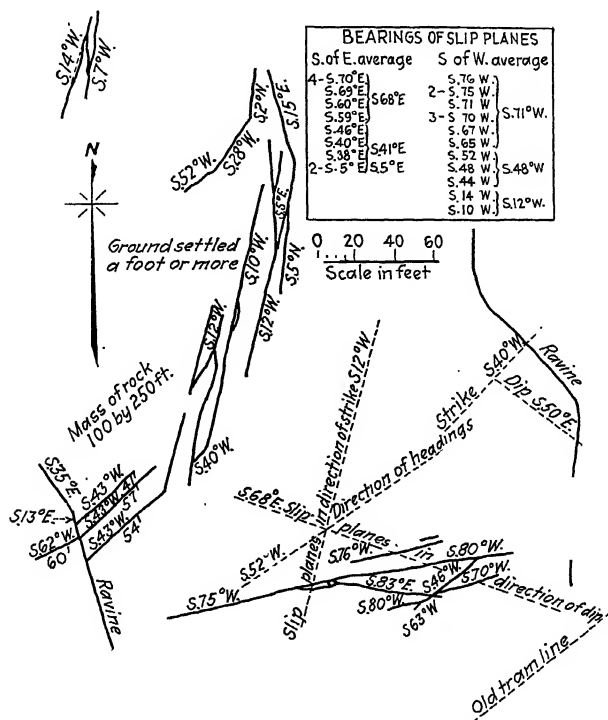


FIG. 16.—MAP OF SURFACE CRACKS, SHOWING BREAKS FOLLOWING SLIP PLANES AND THE MEAN BETWEEN WORKINGS AND SLIP PLANES.

surface and, with more extensive failure of mine supports under greater depth of cover, producing subsidence.

The plan of surface cracks in Fig. 16 shows an area badly broken by cracks. Some of these, as in the upper part of the plan, follow closely the direction of slip planes, while those in the lower part have developed along a mean between the workings and the lines of slip intersecting the workings at a considerable angle. It is interesting to note that a large "horse," or mass of rock several hundred feet in length and breadth, occurs to the left of the center of the area and in the abrupt turn of the

north and south lines of slip planes and those running about east and west. Further, the triple change in bearings of slip planes from the north to the west, in the upper part of the area, shows a circling action about a slight settlement of the surface. Other divergences of lines of cracks from slip planes are due to irregular breaking and settlement around stub pillars.

That subsidence of the surface is taking place is evident throughout limited areas, as shown in Fig. 17. The settlement shown in Fig. 17 is the most extensive observed in the district and extends for about 300 ft., and has a displacement of 6 ft. Settlement will undoubtedly assume greater proportions in the future, but aside from the railroad right-of-way there is little cause for concern from failure of the surface, as the land is not suitable for agricultural purposes and has no buildings of conse-



FIG. 17.—SUBSIDENCE OF GROUND ALONG RAILROAD.

quence. Further, as mining operations have extended for considerable distances beyond the railroad, no precautionary measures can now be taken to protect the surface except to stop the robbing of pillars and to reenforce if possible such supports as may remain.

SURFACE PROTECTION IN FUTURE MINING OPERATIONS

As it may be desirable and necessary, in the future, to protect the surface from subsidence resulting from mining operations, particularly when development by vertical shafts becomes the practice, the means to be employed should be considered. While it is not known how regular the failure of top rock is, it is likely that the condition observed for considerable distance in the top rock will obtain for much greater distances, because the area and height of disturbed ground extends both horizontally and vertically with the failure of pillars between adjacent stopes.

In Fig. 8 is shown how the height of caving and broken ground may be increased by the failure of pillars between stopes, the caved areas above the respective stopes coalescing. The failure of top rock is prob-

ably limited approximately as shown. The breaking and fracturing of cover to a height several times that of extensively caved single stopes, as shown in Fig. 9, has been established by observation. Local conditions of occurrence and arrangement of top rock may modify the character and extent of the failure of cover, but, in general, the facts are established as far as the present character of operations is concerned.

When protecting the surface by leaving supports properly located in the workings, two facts may be considered as the controlling factors involved; namely, the angles of break in caving top rock and the horizontal and vertical directions assumed by slip planes. The dip of the orebed, an important factor, is involved in the angles as determined and need not be considered further.

The surface to be protected against disturbance and failure having been determined, the position of the supporting and protecting pillar or pillars in the workings below remains to be decided. In Fig. 18 is shown a vertical section parallel with the dip of the orebed, but the bed is assumed to be horizontal. In the application of the vertical angles of slip planes and break in top rock it is preferable to choose the angles for the particular locality concerned rather than to use the average, which, though close, may lead to errors in location especially when the depth of cover is considerable; see Fig. 5 for selection of proper angles.

As there may be considerable variation in the bearing of slip planes and consequent breaks on the surface, it is not easy to determine the position of pillars that will provide adequate protection along horizontal lines. However, as the direction taken by the workings is the controlling factor in the horizontal direction of surface breaks, it is safe to be guided by them; observation has shown that lines of failure along slip planes may be definitely checked and turned aside by relatively large areas of pillars or unmined ground. This condition is much more pronounced with considerable depth of cover, while with light cover there is less danger in damage to pillars, particularly with those of larger cross-section. The width of stopes and pillars and effect of angles of dip and break in top rock are shown to advantage in Fig. 18, which together with the diagrams, Fig. 5, gives the data necessary for the application to localities.

RÉSUMÉ

The failure of top rock is the result, in large part, of the presence of jointing or slip planes that cut not only the orebed but the associated formations above and below it. The number and prominence of the slip planes have a profound effect on the strength of the formations in both horizontal and vertical directions. Furthermore, the slip planes parallel with the strike of the orebed and the breaks following them in the top rock are practically vertical, assuming verticality from smaller angles in the orebed.

Failure of the overlying formations originates in the tops of caving stopes, spreading laterally through the collapse of pillars. While the fracturing of top formations is determined by position of stopes and vertical slip planes, the lateral extension of breaks involves other factors as the bearings of one or more lines of slips. Assuming two slip planes of prominence, one roughly paralleling the strike or direction of workings and the other making a much larger angle with the lines of workings, the resultant line of break on the surface approximates a mean, or bisector, of the angle of intersection.

Aside from the effect of caving stopes on fracturing the top formations, the slope of the ground has a marked influence on the formation of surface breaks. Masses of ground weakened by fracturing of cover by caving

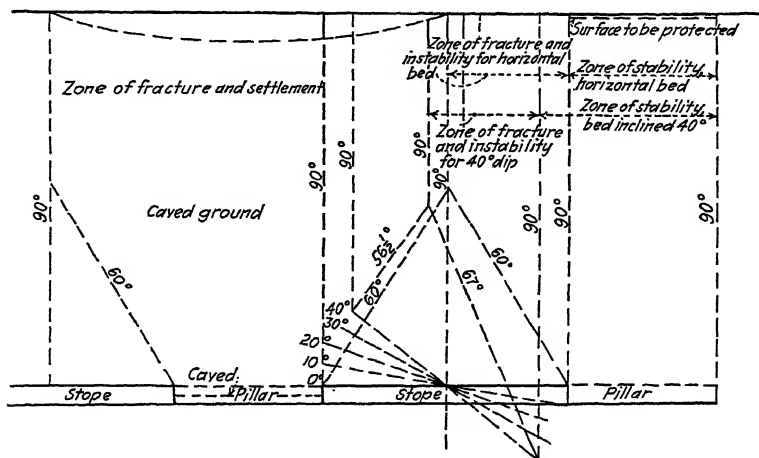


FIG. 18.—METHOD OF DETERMINING LOCATION OF UNDERGROUND SUPPORT FOR PROTECTION OF SURFACE.

stopes break across sloping ground, and by an overturning action develop cracks, the presence of slip planes contributing to such action.

Cave-ins reaching the surface occur only under slight cover, while fracturing and settlement of the surface represent more extensive caving underground and particularly under greater depth of cover. The relative extent of caving in stopes, both horizontally and vertically, can be determined from stope intervals and angles of break in top rock. It is possible, therefore, to determine quite accurately the degree of disturbance attained by caving stopes.

Additional protection for valuable surface can be secured by the application of facts established, such as the angles of break of top rock with given dips of orebed, coupled with similarly known direction of slip planes controlling and influencing vertical and horizontal breaks.

SUGGESTIONS FOR FURTHER STUDY

The statements here set forth are the result of many observations on the occurrence of the orebed on Red Mountain and particularly on jointing or slip planes in the orebed and associated formations. Observations were made in the mine workings, on the surface, and as far as was possible in the formations overlying the orebed.

An area of approximately 12 miles long and $\frac{3}{4}$ mile wide was carefully examined, including much of the old workings and considerable of the more recent; while the whole southeastern slope of Red Mountain was carefully gone over and observations made. Throughout this area, the cover varied in thickness from 50 ft., at the top of the mountain near the outcrop, to 500 ft., in Shades Valley beyond.

Extended observations in old worked-out areas, where failure of top rock can be seen in all stages of development, have revealed numerous examples of natural and automatic filling of stopes and checking of caving. It is plain that any means whereby the caving rock can be broken and indiscriminately mixed, thus occupying the maximum space possible, will contribute to the desired results. Examples of stopes that have been filled by top rock falling upon obstructions, as props, are numerous. The fall of top rock *en masse* and often with slight change in volume simply shifts the open space from a lower to a higher level without tending to check caving; but by breaking up the caving top rock as it falls, either by causing it to fall irregularly or upon rock-filled cribs of old timber or other artificial obstructions, the desired results may be secured. The same results are obtained from natural stoping, which by the nature of the failure puts the caving rock in the best possible condition as a space filler, see Fig. 7.

DRAINAGE OF MINES

This involved study of: drainage districts established; occurrence of ore; conditions affecting drainage; source of mine water; effect of mining on drainage; surface, subsurface and ground-water drainage; exclusion of water from mines; measurement of streams, and suggestions as to remedial measures to be taken.

Pumping records for the mines give interesting and corroborative evidence regarding the relation of precipitation and run-off to the water entering the mines.

Drainage Districts.—The drainage area lies between the outcrop of the Big Seam on the north slope of Red Mountain and Little Shades Creek, paralleling the mountain, and includes the actively operated mines of the district. It comprises all the surface overlying and adjacent to the mines and is that part of Shades Valley that is intimately related to drainage of the mines. The main district is subdivided into six areas

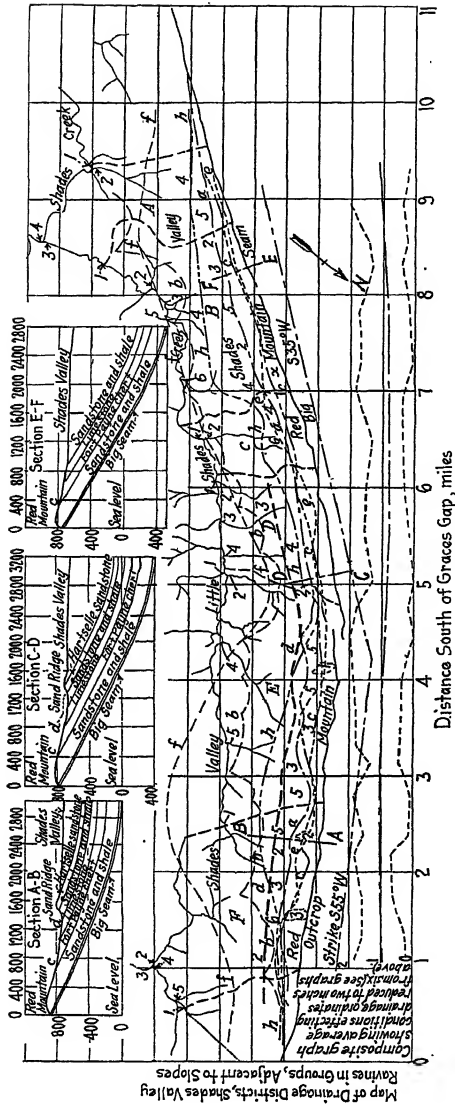


FIG. 19.—MAP OF DISTRICT, SECTIONS THROUGH RED MOUNTAIN AND SAND RIDGE, AND GRAPHS SHOWING CONDITIONS AND FACTORS AFFECTING DRAINAGE IN THE IRON ORE MINES.

A, B, C, D, E and F. Drainage districts. Numbers in districts show ravines adjacent to stopes; those along creek are gaging stations.

a, Approximate limit of abandoned workings; b, limits of workings in Shades Valley; c, outcrop of Fort Payne chert; d, outcrop of Hartselle sandstone; e, present limit of fractures; f, approximate lower limit of faults; h, approximate upper limit of faults.

adjacent to certain groups of mines, the streams in these subdistricts receiving the run-off and discharging into Little Shades Creek with the exception of the two end areas which divide the drainage between Shades and Little Shades creeks. (Fig. 19.)

The areas of the subdistricts together with the area of the entire drainage districts are: District A, 1.24 sq. mi.; district B, 1.82 sq. mi.; district C, 0.67 sq. mi.; district D, 1.12 sq. mi.; district E, 2.49 sq. mi.; district F, 2.19 sq. mi.; total area, 9.56 sq. mi.

Jointing or Slip Planes.—The ore bed is cut by numerous joints ("slips"), extending for long distances with slight change in direction.

Whereas the majority of the joints are "tight," numerous open-joints or slips occur in the recent as well as the old workings. Many admit but little water, a wetness of the roof alone showing a small but constant supply of water.

Cross-bedding in Ore Bed and Top Rock.—Cross-bedding in both ore and top rock is similar in effect to jointing but is less important as it is limited to individual beds. It is mainly a source of weakness of the beds of sandstone and slate, causing them to break readily and open.

Bedding Planes in Top and Bottom Rock.—Owing to the thinly stratified formations over and underlying the ore bed, the bedding planes permit passage of water mainly along rather than between them. The drag of the formations adjacent to and often for considerable distances from extensive faults, may seriously weaken, if not loosen, the bond between the beds, permitting water to be transferred, particularly on the dip.

The heaving of the bottom rock along the lines of the stopes loosens the rock on the bedding planes and often causes them to break down. The forming of trenches and the driving of underpasses, rock slopes and tunnels indicate water moving in the rocks underlying the ore bed.

CONDITIONS AFFECTING DRAINAGE

Folding and Faulting of Strata.—The ore bed has been warped by folding causing a rather wide range in dip—12° in the northern part of the district and 35° in the southern part. Faulting has generally accompanied the folding, and is most pronounced along minor and counter folding, although the larger faults are apparently independent of all other disturbances. Weakening of the ore bed and associated formations is a direct result of folding and faulting and contributes appreciably to the wetness of the mines in certain localities.

Faults are distinct lines of weakness. Many are tight the greater part of their extent; on the other hand, they may admit considerable water. Secondary faulting both above and below the main faults, or the opening of jointing planes in the top and bottom rock, may result in a bad condition. The importance of faults and faulted areas as vehicles for the conveyance of water is demonstrated where sections of workings

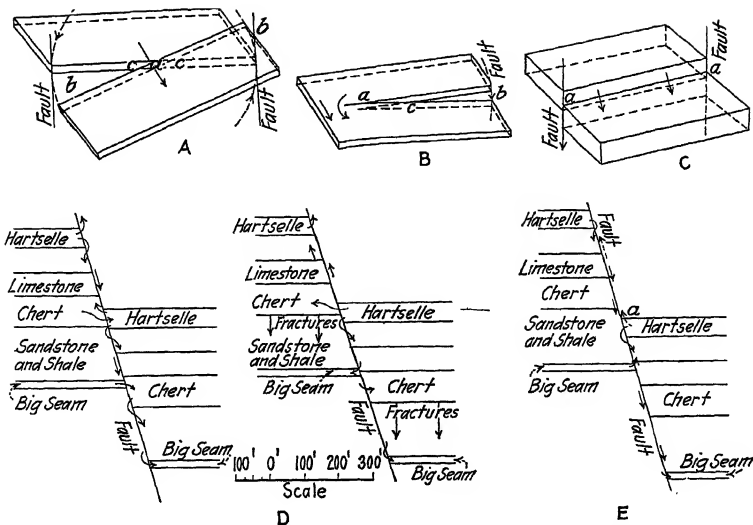


FIG. 20.—SKETCHES SHOWING TRANSFER OF WATER BETWEEN PERVIOUS AND IMPERVIOUS FORMATIONS AS AFFECTED BY FAULTING AND FRACTURING. A, B and C show effect of faulting on movement of water down a slope. D and E show effect of faulting on vertical movement of water in overlying formations.

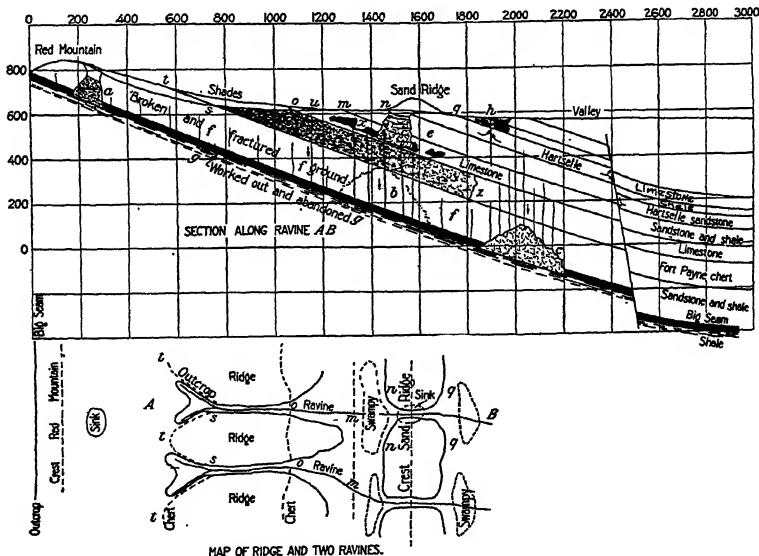


FIG. 21.—IDEAL SECTION SHOWING SOURCE AND MOVEMENT OF MINE WATER.

previously rendered wet by water entering through a fault suddenly become dry when the fault is opened at a lower level. (Fig. 20.)

Drill holes may open water pockets in joints otherwise dry, or release water in joints not exposed by mining operations. Holes may also permit water to escape freely from areas of broken ground resulting from local disturbance of the rocks or fractured by excessive use of explosives; for example, upsets where the rock is often badly broken by heavy shooting.

Miscellaneous Conditions Affecting Drainage.—Among the formations above the ore bed are several beds high in lime that are cavernous. It is evident from the numerous cave-ins along the lines of outcrop and the regularity of their occurrence, that the beds are potential sources of mine water and are undoubtedly contributing to the present supply through fractured top rock. In Fig. 21 are shown two such lines of cave-ins, one on either side of Sand Ridge, which condition exists throughout the district.

SOURCE OF MINE WATER

It is desirable to distinguish between surface, subsurface and ground waters. These sources of water are supplied from rainfall and vary according to the precipitation, but the subsurface and ground waters are much more uniform in flow than the surface waters and yield a fairly constant volume throughout the year. The surface, subsurface and ground waters may be approximately equal in volume and differ only as to time, which may throw the preponderance of influence to surface water. However, the source of supply of ground water may be widespread and distant, furnishing much larger quantities than the surface and subsurface waters and with a flow much more uniform.

Surface Water.—Rain water that runs off the earth's surface is called surface water, that which sinks into the accumulations of loose surface material is subsurface water. It is difficult to distinguish between the two sources as the moving waters mingle and overlap. Surface water enters the underground workings directly, whereas subsurface and ground waters pursue circuitous paths and may require hours, days and even weeks. To control surface water it must be excluded or promptly ejected.

Subsurface Water.—Surface water enters the soil or loose surface material at a rate depending upon the character and density of the material and its degree of saturation, which in turn also affects the amount of surface water, the run-off from a given area showing marked variations depending upon the interval between rains. Subsurface water moves downward and laterally in the direction of the slope, which is the controlling factor in the quantity of subsurface and ground waters that results from a given rainfall.

Ground Water.—Both surface and subsurface waters contribute to the supply of ground water, the latter providing probably the greater amount. Water that enters the ground and which does not escape into streams continues its downward course filling up porous formations and forming the great body of water known as ground water. Its level is usually that of the streams draining the district, but may be considerably higher, due to the resistance of movement through rock masses in the adjacent higher ground, the topography of the ground and the character of the rocks having a pronounced influence. The profile of the ridge of Red Mountain and the water-level are shown in Fig. 19, the latter being fairly accurately located except where movement of ground has caused it to fall through the water having drained off.

Porous Formations above Ore Bed.—The range in thickness and average thickness of the formations, in order of occurrence from below upward, are given in Table 1.

Owing to its proximity to the Big Seam, the Fort Payne chert is considered the most important from the standpoint of mine drainage. It is a cherty limestone, rather than a high-lime chert. The chert bed proper and the limestone (Fort Payne) overlying it were formerly included under the head of Fort Payne chert, but owing to the purity of the latter it has been separately designated as a limestone.

TABLE 1.—*Occurrence and Thickness of the Big Seam and Associated Formations*

Formation	Range in Thickness, Ft.	Average Thickness, Ft.
Limestone.....	41 to 90	60
Shale.....	7 to 48	34
Hartselle sandstone.....	11 to 143	60
Shale.....	55 to 215	80
Fort Payne limestone.....	0 to 127	75
Fort Payne chert.....	29 to 206	100
Slate and sandstone.....	115 to 200	150
Big Seam.....	15 to 22	20
Total thickness.....		579

The porosity of the Fort Payne chert is due to solution cavities rather than to intergrain space. As a water-carrier it depends upon the extent of the leached zone. It is variously estimated that the leached zone of the chert bed has a width, as measured on the dip, of 1500 to 2000 ft. or a vertical depth of approximately 400 to 600 feet.

The Fort Payne limestone having much less chert and more lime has undergone a greater change through solution than the chert bed and is cavernous, as shown by cave-ins along the outcrop.

The next porous formation, in order of importance, above the Fort Payne limestone is the Hartselle sandstone, about 405 ft. above the Big Seam and 80 ft. above the limestone. It has an average thickness of 60 ft., but ranges from 11 to 143 ft. It forms Sand Ridge which is close to 1500 ft. from the crest of Red Mountain. (Fig. 21.)

The Hartselle sandstone is moderately hard but has a high porosity, absorbing and discharging water freely. Being higher to the northeastward, the pitch of the sandstone is to the southwest, and this is made more pronounced by the change in dip of the formations, particularly in the mining district, giving a decided inclination of the beds including the Hartselle sandstone from the northeast to the southwest, and parallel with Red Mountain. (Fig. 19.) Any water contained in the Hartselle sandstone must move by gravity from the higher to the lower portions, consequently the sandstone bed enclosed as it is in impervious beds of slate must bear artesian water, the source of which may be many miles to the northward.

The last porous formation of interest in connection with mine drainage is a bed of limestone averaging 60 ft. in thickness and occurring about 35 ft. above the Hartselle sandstone, the intervening material being shale. This, like the Fort Payne limestone, is cavernous and its outcrop is marked by sinks or caveins on the south side of Sand Ridge.

EFFECT OF MINING ON DRAINAGE

The failure of top rock and the consequent caving of stopes is probably responsible for the greater part of the water that enters the mines. The following kinds of breaks in top rock are typical; caved stopes, cave-ins, surface fractures or cracks, and sinks or zones of settlement.

Caved Stopes.—The first effect of failing top rock is the collapse of the roof of stopes, which may ultimately extend to considerable height. When several stopes are united by caving a large area of top rock may be badly broken and extend vertically for several hundred feet. Failure usually takes place along lines of weakness such as joints, thus adding to the height and lateral extent of the area affected.

Cave-ins result from caving in stopes, which extend to the surface but rarely occur beyond 500 ft. from the outcrop, consequently they indicate breaks in depth of cover up to 250 ft., and possibly more.

Surface Cracks.—Numerous surface cracks occur on the south slope of Red Mountain, along the courses of ravines and on the flats at the foot of the slope of the mountain. These open and close in order of their occurrence, due probably to a downward and backward swing in direction of the slope of the surface. This has a direct bearing upon drainage, for ground at one time badly broken by cracks may later become so tight as to exclude much if not all water coming to it.

Sinks.—The settlement of the surface over considerable areas, usually larger than cave-ins, occurs where the cover is much thicker than in the case of cave-ins. Settlements usually indicate a zone of fracturing and may be of greater importance than cave-ins or cracks owing to the more extended area.

SURFACE, SUBSURFACE AND GROUND WATER DRAINAGE

Surface drainage is influenced by vegetation, character of soil and disintegrated surface materials, slope of surface and presence of cave-ins and surface fractures, etc., the last three factors being most important. Observations made on the run-off in ravines on the south slope of Red Mountain during two rain storms of different amounts of precipitation and duration show that the run-off varies directly with the degree of fracturing of surface. There were, however, large run-offs in badly-fractured areas, due to the stream beds being filled with silt, the cracks being fairly effectively closed thereby.

The significant fact brought out by these observations is the relation between intensity of precipitation and duration of a storm—a small precipitation for a short period being more effective in producing a run-off than a heavier precipitation for a longer period.

Surface water entering the natural lines of drainage from unbroken ground has not passed the point where it can be intercepted and diverted underground to the mines. In many instances the water, running in the ravines on the south slope of Red Mountain, even at times of large run-offs, is unable to cross the outcrop of the Fort Payne chert without being wholly engulfed.

Subsurface Drainage.—It is difficult to measure accurately surface and subsurface waters and thereby to distinguish between them, the difficulty increasing with amount of loose surface materials and the slope of the surface. Surface water discharges rapidly, but some time is required before the subsurface water enters lines of drainage and reaches its maximum flow, and particularly a long period for the completion of the discharge. It is evident then that surface water ordinarily reaches its maximum run-off and may be in its decline before subsurface drainage really begins, after which the two discharge together, building up to a maximum and then diminishing slowly until all subsurface water that will enter the lines of drainage has been exhausted.

Ground Water Drainage.—A relatively large part of the subsurface water of the entire district enters the porous and broken formations exposed on the south slope of Red Mountain and which are traversed by numerous lines of drainage. This water ultimately reaches the mine workings.

The control of the movement of ground water by faults is shown in Fig. 20. The over-lap of the formation from the line *aa* downward gives

the effective area through which water may pass; in the latter, with the rotary fault, the effective area is cc and becomes less as the displacement becomes greater, while in the case of a fault tailing out as in A the movement of water must be around the fault proper.

Here conditions of drainage are given where the Fort Payne chert is leached and therefore porous, in which case it can give to or take from the Hartselle sandstone according to the relative breaks existing between the two. In E , a condition exists where the Fort Payne chert is not porous and must therefore be considered as an impervious bed;

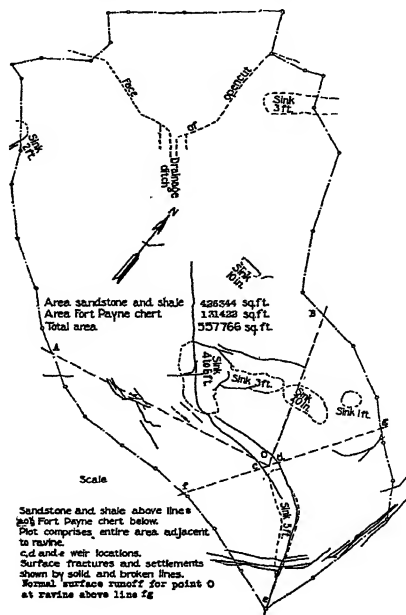


FIG. 22.—DRAINAGE AREA NO. 3 IN WHICH SURFACE FRACTURES AND SETTLEMENTS CROSS THE PROPERTY APPROXIMATELY NORMAL TO THE RAVINE.

water entering the workings in the Big Seam is then supplied by the Hartselle sandstone.

In general then, meteoric water enters the mine workings as surface water directly through cave-ins, surface cracks and sinks, and as sub-surface and ground water through the breaking of porous formations that are water-bearers, which together with the geological occurrence of formations, presence of faults, etc., are shown in the vertical section, Fig. 21.

The map at the bottom of Fig. 21 shows two ravines and their relation to the outcrop of the Fort Payne chert on the ridges at t and in the ravine at s and o , also the outcrop of the limestone bed above the chert at o

and *m*, the outcrop of the Hartselle sandstone being shown at *n* and *q*. The occurrence of cave-ins and caved stopes are shown at *a*, *b*, *c* and *e* and fractures in top rock at *f*. The leached outcrop of the Fort Payne chert is represented by shaded ground and the limit is given at *z*. Movement of water along fault planes, in artesian wells and in top rock, is shown by arrows.

The entry of water through surface fractures and broken rock at the top of a caved stope is shown in Figs. 22 and 23.



FIG. 23.—WATER ENTERING MINE THROUGH BROKEN TOP ROCK.

EXCLUSION OF WATER FROM AND ITS CONTROL IN THE MINES

It is obviously impossible to prevent rain water from entering the outcrops of porous formations, consequently the only source of mine waters that can be controlled from the surface is that entering the mines directly through cave-ins, surface cracks and other disturbances of formations overlying the ore bed worked, and that reach to the surface.

The broken beds of ravines and ditches, through which surface water drains and mine water is discharged should be either abandoned as means of conveying water or rendered non-leakable. The natural silting of such water-courses, by the wash of clays and sands along their courses, provides striking examples of what may be done.

Much water is prevented from entering the mines through diverting ditches judiciously placed and their extended use is amply justified.

Flumes may be used, and are to a limited extent, in conveying water over fractured ground and relatively large areas that have settled. So far as present practice is concerned the means given above for the exclusion of surface water from underground workings are all that are employed.

The exclusion of surface waters from the mines is important, but that of ground water is of vastly greater importance because of its source and large volume. The water-bearing formations are capable of becom-

ing constant sources of water, while the water-filled caverns can probably be emptied quickly, although the damage done temporarily may be large.

Weakening or failure of the formations overlying the ore bed, as through extensive collapse of roof in workings, may cause breaks to occur that may extend to the water-bearing formations, opening up channels through which drainage may take place. This supply of ground water will increase with an increase in area of caved stopes until it is sufficiently large to reduce the level of such water in the water-bearing beds.

The control of water, once having reached the workings, is readily accomplished by the use of diverting ditches and pipes, catch-basins,

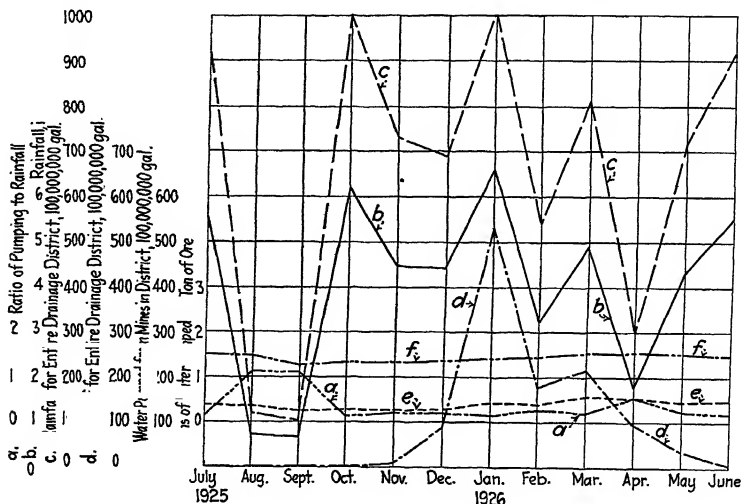


FIG. 24.—RAINFALL, RUN-OFF AND WATER PUMPED FOR A 12-MONTH PERIOD, ALSO TONS OF WATER PUMPED PER TON OF ORE RAISED, WITH RATIO OF PUMPING TO RAINFALL. AREA OF DRAINAGE DISTRICT 266,562,000 SQ. FT., OR 9.561 SQ. MILES.

stoppings and sumps, which can handle large volumes of water and may equalize the flow to sumps.

Mines are usually grouped together according to slopes, two, three, and four being common groupings. The inflowing waters from all sources are collected by and delivered to auxiliary sumps by natural flow or through ditches, and in turn conducted by pipes from the auxiliary sumps to a main sump located adjacent to one of the slopes.

MEASUREMENT OF STREAMS

Rainfall and Run-off.—The monthly rainfall in the Birmingham district, as reported by the local Weather Bureau, is shown in Fig. 24. The run-off was obtained by rod and weir measurements on Shades and Little Shades creeks and their tributaries. The graphs show the

effect of cumulative rainfall in starting run-offs, particularly following periods of dry weather. The record of run-off is a fair average but does not show the discharge of the streams during floods, because reliable data cannot be obtained.

Surface and Subsurface Drainage.—Rod gaging was employed on all streams and where possible was supplemented by weir and current meter check tests. The most careful and detailed observations were made, particularly as the stream beds are irregular and during the summer months the current is slow and in some cases decidedly sluggish. Weirs were also employed in measuring pump discharges and to secure the relative amounts of surface and subsurface drainage.

The most interesting and valuable data relative to surface and subsurface drainage were obtained from weir measurements on drainage areas adjacent to ravines on the south slope of Red Mountain, where no water flows except during storms or wet weather. (Table 2.)

Similar comparative tests were also made in ravines and drainage ditches in order to determine the loss of water in transit over formations of different character and condition and are shown in Table 3 and Fig. 22.

SUGGESTED REMEDIAL MEASURES

Observations and measurements have been made covering a wide range of conditions in the Birmingham district, and definite and concrete facts have been established relative to the occurrence and amount of water falling on the surface and that reaching the underground workings. In view of the information secured the following suggestions may be offered looking toward the solution, in part at least, of the problem: (1) The use of carefully placed diverting ditches and their maintenance in good condition; (2) the silting of stream beds and all ditches crossing loose surface materials and outcrops of porous formations; (3) the use of flumes and pipes to conduct water over broken ground; (4) checking the caving of stopes and thereby preventing ground movement resulting in surface fractures; (5) isolating, if possible, wet areas in the workings; (6) employment of hydraulic rams to eject part of the water from the mines, and (7) the employment of caving systems within panels.

There are two remedies for such adverse conditions: one is to silt the ravines or ditches by providing clay for the water to distribute; the other way is to use wooden or metal troughs or pipes for conveying water across broken ground—slip-jointed pipes would be most effective. The silting of stream beds is being done naturally, thus demonstrating that ravines with broken or porous beds can be effectively treated in like manner.

As to underground conditions, the most important is the entrance of water through broken top rock. Top rock will not break unless stopes are permitted to cave indiscriminately. The remedy is the prevention of excessive caving by filling the old stopes with caved rock, when signs of failure occur.

SUBSIDENCE AND ITS RELATION TO DRAINAGE

TABLE 2.—*Results of Observations on Drainage Areas Giving Comparative Data Relative to Rainfall, Run-off and Effect of Character of Formations and Conditions of Surface*

Drainage District No.	Rainfall				No. 1 Weir						No. 2 Weir §	
	Inches	Duration		Rate of Precipitation per Hour, Inches	Area Drained Sq. Ft.	Rainfall		Run-off			Area Drained, Sq. Ft.	Rainfall Cu. Ft.
		Hr.	Min.			Cu. Ft.	Gal.	Cu. Ft.	Gal.	Per Cent.		
1	1.86	5	45	0.32	360,069* 77,513†	67,825	514,812	0	0	0	360,069* 8,168†	57,077
1	1.52	5		0.30	360,069* 77,513†	55,429	414,609	0	0	0	360,069* 8,168†	46,045
2	0.48		40	0.72	101,068* 148,117†	9,967	74,553	0	0	0	101,068†	4,043
2	0.29		8	2.18	101,068* 148,117†	6,022	45,045	0	0	0	101,068†	2,443
2	0.47		10	2.82	101,068* 148,117†	9,760	73,005	0	0	0	101,068†	3,959
2	0.23		5	2.76	101,068* 148,117†	4,801	35,911	0	0	0	101,068†	1,937
2	0.20	1		0.20	101,068* 148,117†	4,153	31,064	0	0	0	101,068†	1,685
x	1.86	5	45	0.32	No weir measurements were taken in this area as there were no run-offs at the weir locations due to badly broken ground.						316,245* 69,870†	59,848
					316,245* 87,706†	62,612	468,338	3,811.4	28,509	6.08		

No. 2 Weir§				No. 3 Weir						Increase or Loss between Weirs, Per Cent.		Date (1926)
Rainfall	Run-off			Area Drained, Sq. Ft.	Rainfall		Run-off			Nos. 2 and 1	Nos. 3 and 2	
Gal.	Cu. Ft.	Gal.	Per Cent.		Cu. Ft.	Gal.	Cu. Ft.	Gal.	Per Cent			
426,936	1,223.7	9,153	2.14	360,069*	55,811	417,466	3,359.1	25,126	6.0	-100	-64.2 distance 62 ft.	March 30
348,905	6,821.4	51,024	14.6†	360,069*	45,610	341,163	11,515	86,132	25.0	-100	-40.7 distance 62 ft.	March 10
30,242	27.2	203	0.67							-100		May 2
18,274	67.5	505	2.76†							-100		May 3
29,613	165.2	1,236	4.18		(Two weirs only)					-100		May 19
14,489	99.0	841	5.11†							-100		May 19
12,604	22.1	165	1.31†							-100		May 19
447,663	1,994.4	14,918	3.34		(Two weirs only)					+91.1		March 30

NOTE: No run-off occurred at the time of making observations on any of the areas, owing to broken ground or exposed outcrop of Fort Payne chert, also to small or moderate rainfall. Two storms producing definite run-offs occurred during the period of observation on the areas, but no accurate or complete records were obtainable at the time.

* Ferruginous sandstone.

† Chert.

‡ Soil saturated wholly or in part.

§ Weir No. 2 was placed 62 ft. below No. 3, at a point beyond the contact between the sandstone and shale and the Fort Payne chert, in order to get information on flow over chert outcrop. See c Fig. 11.

TABLE 3.—*Loss in Flow of Steams through Ditches Traversing Different Kinds of Formations and under Different Physical Conditions*

Designation	Formations Traversed by Ditch	Length of Ditch, Feet	Grade of Ditch, Per Cent.	Total Quantity of Water				Difference in Amount of Water		Total Increase or Loss between Weirs, Per Cent.	Increase or Loss per 10 Ft.									
				Upper Weir		Lower Weir		Cu. Ft.	Gal.		Cu. Ft.	Gal.								
				Cu. Ft.	Gal.	Cu. Ft.	Gal.													
A	Clay, hard and dense.....	790	1.7	(1)	2,668.4	19,960	2,690.4	20,124	+	21.6	+	164	+	0.27	+	2.1	+	.0101*		
				(1)	3,785.4	28,315	3,692.5	27,620	—	93.0	—	695	—	2.5	—	1.89	—	14.2	—	.0508
				(2)	2,297.3	17,184	2,048.3	15,321	—	249	—	1,863	—	10.8	—	5.07	—	37.9	—	.2200
				Av.	3,041.4	22,749	2,870.4	21,471	—	171	—	1,279	—	5.62	—	3.48	—	26.0	—	.1143
B	Clay, hard and dense. Low point in ditch through which water moved slowly.	491	2.6	(1)	12,850.4	96,121	11,193.3	83,726	—	1,657.2	—	12,395	—	12.89	—	95.7	—	.1998		
				(2)	8,513.5	63,681	7,555.6	56,516	—	957.9	—	7,165	—	11.25	—	44.8	—	111.0	—	.1743
				Av.	10,681.9	79,901	9,374.4	70,121	—	1,307.5	—	9,780	—	12.25	—	20.3	—	152.0	—	.1900
				(1)	1,126.1	8,423	1,025.5	7,670	—	100.6	—	753	—	8.9	—	2.74	—	20.5	—	.2422
D	Sandstone and shale, 267 ft. Fractured chert bed, 249 ft. Chert, 100 ft.....	367	4.4	(1)	1,016.6	7,604	912.5	6,825	—	104.1	—	778	—	10.2	—	2.84	—	.5740		
				(2)	593.6	4,440	477.5	3,572	—	116.1	—	368	—	19.6	—	3.16	—	23.6	—	.6340
				Av.	912.1	6,822	805.2	6,022	—	106.9	—	800	—	11.72	—	2.91	—	21.8	—	.3200
				(1)	1,424.9	10,658	1,086.6	8,128	—	338.3	—	2,530	—	23.75	—	5.5	—	41.0	—	.3825
E	Leaky flume, 371 ft. Fractured chert bed, 249 ft. Entire distance over broken chert bed	620	3.9 flume 4.4 ditch 3.9 flume 4.4 ditch	(1)	9,941.4	74,302	7,423.2	55,525	—	2,518.3	—	18,837	—	25.30	—	40.6	—	.408		
				(2)	5,083.2	42,510	4,251.9	31,827	—	1,428.3	—	10,684	—	25.13	—	23.1	—	172.5	—	.4055
				Av.	7,512.3	58,406	6,097.6	46,176	—	1,715.3	—	17,761	—	25.22	—	31.8	—	242.2	—	.4067
				(1)	3,360.8	25,195	3,411.5	25,521	—	51.2	—	386	—	1.6	—	2.21	—	16.7	—	.0672
F	Clay filled with chert fragments. Ground sloping to ditch on either side.	231	4.4	(1)	10,964.3	82,013	10,886.1	82,170	—	21.8	—	163	—	0.2	—	40.94	—	.0087		
				(2)	7,162.3	53,574	7,198.8	53,848	—	36.5	—	275	—	0.51	—	11.9	—	11.9	—	.0220
				Av.	9,063.3	67,793	9,042.5	67,954	—	20.6	—	169	—	0.36	—	11.42	—	11.42	—	.0220
				(1)	3,360.8	25,195	3,411.5	25,521	—	51.2	—	386	—	1.6	—	2.21	—	16.7	—	.0672

* The figures given in this column are not averages of the percentages given, but are calculated from total quantities.

DISCUSSION

[The following is discussion of the paper, "Mine Subsidence in Red Iron Ore Mines of the Birmingham District, Alabama," presented at the New York Meeting, February, 1925.]

G. S. RICE, Washington, D. C.—In view of the breaks to and through the Fort Wayne chert that already let water into the mines, would it bring in materially more water in the future to extract the pillars and bring down the surface evenly? There is a limit to the amount of rainfall that would drain into the mines, as the surface overlying the leached chert is sloping, and if the stream coming from farther up the valley can be diverted the question is would there be much more water coming through present breaks to be pumped out of the mines? In any case, by increasing extraction, would not the recovery of 30 to 40 per cent. of iron ore more than pay for the additional pumping? If there were no breaks to the surface I could understand the advisability of leaving in pillars. It may be that this question has been considered, but if not, I should think that a survey of the matter would be worth while.

W. R. CRANE.—The water problem is a serious one and probably will always be handled but with a constantly increasing operating cost. Instead of bringing in filling, to hold up this top and prevent the angles, that we have determined, from developing and consequently the height of stope that would result, my suggestion is to provide a filling of rock. In ordinary mines, if we can cause the top to break up thoroughly when it falls, its bulk will increase from 35 to probably 40 per cent. The roof is going to fall anyway. The operator should not systematically go through the old stopes and cave material all the way along, but where caving is started, barriers should be built, for instance, with shale, or the bottom should be blasted up or the top shot down here and there. Then when the top falls, instead of its falling like a loose wall it will break up and in a short time the space will be filled. We prevent those structures from falling into the Fort Wayne chert.

L. S. CATES, Salt Lake City, Utah.—What is the ratio between the value of the pillars left and the pumping space?

W. R. CRANE.—From now on 70 per cent. will be left permanently. In the old workings, they sometimes went as high as 85 per cent. There is a case where rock operations have caused serious trouble.

L. S. CATES.—Is there no chance to work out a draining scheme?

W. R. CRANE.—They have a fine draining system, using diverting ditches on the surface, but the whole surface is becoming fractured. The mines are linked together by ditches and pipes which feed to large pumping stations, so that operators are doing all they can and doing it very efficiently. Every time a crack comes in a new area we get more water.

G. S. RICE.—The valleys are not cut low enough to get to the tunnel?

W. R. CRANE.—It may be a matter of too many tunnels. Where faults occur bringing the Fort Wayne chert down at the orebed, draining has taken place naturally, because all the water that comes into this mine comes down against Fort Wayne chert.

L. S. CATES.—What is the average pumping expense?

W. R. CRANE.—I cannot tell. Some of the mines are pumping 1,500,000 gal. a day; some are not. One operator said they were pumping 15,000 gal. a min. I do not know what the operating cost is; it is rather high. In wet seasons, we get con-

siderable water in a comparatively short time. Even though we are handling all the water that comes in from the Fort Wayne chert, as far as it cracked, when this wet spell comes pumping is greatly increased.

[The following is discussion of the paper, "Drainage in the Red Iron Ore Mines of the Birmingham District, Alabama," presented at the New York Meeting, February, 1927.]

E. E. ELLIS, New York, N. Y.—The author deals with the existing mining operations which are close to the surface. The ore in Birmingham is a bedded deposit and all of the rocks are tight. For instance, the porosity of the ore is only about 4 or 5 per cent. on the average, and the moisture content will hardly exceed 1 per cent., and that is characteristic of all the formations.

The problem dealt with has been the drainage in the existing mines, which do not extend over 4000 ft. down the dip, and consequently those mines get the effect of the erosion channels near the surface, ravines and more or less disintegration and decomposition of the rocks. Therefore, the discussion, although it applies to the existing mines, will not have as important a bearing on future mining which will be carried on farther down the dip, because there the formations are tight and what little history we have indicates a small amount of water in the overlying rocks. For instance, the Shannon inclined shaft goes down to the ore about 1900 or 2000 ft. below the surface. That shaft had only a small amount of surface water which was stopped by cementation, and from there on down they had no water whatever in the mine, it was bone dry. The Shannon shaft did get some water by putting a drill hole through a fault, and that water was rather highly mineralized saline water, but I see no expectation of any appreciable water trouble in those mines at greater depth.

E. F. BURCHARD, Washington, D. C.—Faults that cut water-bearing horizons should, however, be regarded as potential sources of water and wherever their presence is suspected conditions should be tested by drilling ahead of the mine workings, if possible, in the Birmingham district.

C. S. BLAIR, Birmingham, Ala. (written discussion).—The author has assembled a great deal of valuable data, but has not emphasized the really important points of danger of water in the mines. He describes in great detail all possible sources by which water could enter mines, including joints, faults, etc., with a description of all the important or supposedly important water-bearing horizons.

It is our feeling that the only really important and dangerous water horizon which has affected the mining to date is the weathered portion of the Fort Payne limestone and chert, and the especially dangerous zone is limited to a belt 500 or 600 ft. wide following the outcrop of the Fort Payne limestone, which is entirely weathered on the outcrop so that practically no outcrops occur, and is weathered to a distance of several hundred feet from the outcrop. This limestone outcrops throughout the field in the low valley just ahead of Red Mountain, and above the Fort Payne chert. This valley is usually followed by small surface streams, and it is our belief that there are numerous underground channels in this limestone horizon through this belt. Practically all heavy water now occurring in the red ore mines has come from too extensive robbing directly beneath this zone. Outside of this general belt the limestone and chert is entirely unweathered and carries no water, and over the main ore-bearing area the Hartselle sandstone is not an important water carrier as it is very fine-grained and usually limy. It was entirely dry in the Shannon shaft.

Some criticism could be made of the figures and diagrams accompanying the paper which are rather hard to decipher, and have been reduced to a scale where they are almost unreadable.

W. R. CRANE (author's reply to discussion).—Mr. Ellis states that the formations are "tight" and that there is little danger of water troubles in future mining operations. A similar statement was made to me by an operator of one of the largest group of mines before I had begun my investigations. A year or so later the same person remarked, "as you know a large amount of water enters the mines through faults," which agrees fairly well with my own observations.

In the Shannon mine considerable water was encountered at a fault zone and while salty at first cleared rapidly. The fact that this area had to be cemented off is sufficient proof that considerable water was encountered and the management was apprehensive that more might be expected. Many mines are dry before they become wet and Shannon was fortunate, for the time being at least, to be able to isolate the wet zone.

If Mr. Blair will refer to pages 12, 13, 17, 24 of *Pamphlet No. 1639-I*,⁴ he will find his objection fully answered; special emphasis was placed on the importance of the Fort Payne chert. However, there is little doubt but that the Hartselle sandstone will be a prominent source of supply of waters entering the mine in future operations. The importance of this "supposed" water-bearing horizon is indicated by the volumes of water flowing from springs and wells at Red Gap, Graces Gap and Halls' spring.

The criticism relative to interpretation of several diagrams is well-founded and is due in large measure to excessive reduction in size of drawings.

⁴ W. R. Crane: Drainage in the Red Iron Ore Mines of the Birmingham District, Alabama. Issued as Pamphlet No. 1639-I, with MINING AND METALLURGY, February, 1927.

OTHER PUBLICATIONS OF THE YEAR

As explained in the Preface the Institute now issues regularly in addition to the TRANSACTIONS in which series this volume is included, a number of other publications. In 1927 these included:

PETROLEUM DEVELOPMENT AND TECHNOLOGY IN 1926; a volume of 956 pages.

PROCEEDINGS OF THE INSTITUTE OF METALS DIVISION, 1927; a volume of 507 pages.

Technical Publications; a series of individual papers grouped and numbered in series.

The YEARBOOK; containing a list of members, officers, committees, official business reports and the constitution and by-laws.

MINING AND METALLURGY; a monthly journal.

Of these the TRANSACTIONS, MINING AND METALLURGY and the YEARBOOK are sent to all members. The special volumes and *Technical Publications* are sent without further request to all members who have indicated for their first preference papers in the groups covered. They are available to all other members on request and month by month those newly issued are indicated in MINING AND METALLURGY, which also contains an abstract of every paper issued by the Institute. Temporary binders suitable for holding each group of *Technical Publications* are furnished at cost, and it is planned to furnish from time to time title and content pages so as to permit binding in groups those not assembled in the established series of special volumes.

For the convenience of members in finding individual papers not reprinted in this volume of TRANSACTIONS there will be found on the pages following complete tables of contents of the special volumes, a classified list of all numbered *Pamphlets* and *Technical Publications*, and a complete index for MINING AND METALLURGY for the year.

Members are invited to notify the Secretary of any changes they may wish to make in their choice of publications.

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1034	Time Factor in Depletion of Mines	J. W. Roberts	New York Feb., 1921	66, 711
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1043	The Health of the Underground Worker	A. J. Lanza	New York Feb., 1921	
1045	Dust-ventilation Studies in Metal Mines	D. Harrington	New York Feb., 1921	66, 272, 282
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1180-M	Deposition of Copper Carbonate from Mine Water	P. D. Wilson	San Francisco Sept., 1922	68, 552
1188-M	Shot-firing by Electricity	N. S. Greensfelder	San Francisco Sept., 1922	68, 216, 232
1191-M	Metal-mine Ventilation in the South- west	C. A. Mitke	San Francisco Sept., 1922	68, 377, 403

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1193-M	The Aztec Mine, Baldy, New Mexico	C. A. Chase, D. Muir	San Francisco Sept., 1922	68, 270
1206-M	Review of Present Status of Drill Steel Breakage and Heat Treatment	F. B. Foley, C. Y. Clayton, H. S. Burnholz	New York Feb., 1923	69, 648, 677
1208-M	Hardness and Heat Treatment of Min- ing Drill Steel Shanks	C. Y. Clayton	New York Feb., 1923	69, 666, 677
1210-M	Liquid-oxygen Explosives at Pachuca	M. H. Kuryla, G. H. Clevenger	New York Feb., 1923	69, 271, 321
1211-M	The Rove Tunnel	M. Mathieu	New York Feb., 1923	69, 248
1224-M	Mining Methods of the Copper Range Co.	W. H. Schacht	New York Feb., 1923	72, 346
1225-M	Mining Methods in the Butte District	W. B. Daly et al.		72, 234
1228-C	Mine-drainage Stream Pollution	A. B. Crichton	New York Feb., 1923	69, 434, 443
1235-M	Mining Methods at the Bunker Hill and Sullivan Mines	H. M. Childs, S. A. Easton	New York Feb., 1923	72, 305
1237-M	Mining Methods of Hecla Mining Co.	J. F. McCarthy, C. H. Foreman	New York Feb., 1923	72, 319
1239-M	Mining Methods of Marquette District, Michigan	S. R. Elliott, J. E. Jopling, R. J. Chennecour, E. L. Derby	New York Feb., 1923	72, 122
1245-C-M	Discussion of Coal and Mining Papers Presented at New York Meeting, 1923			
1251-M	Secondary Enrichment at Eagle Mine, Bonanza, Colo.	C. E. Wuensch	Canadian Aug., 1923	69, 96
1255-M	Discussion of Drill Steel Papers Pre- sented at New York Meeting, 1923			
1258-M	Mining Methods at Bawdwin Mine	A. B. Calhoun	Canadian Aug., 1923	69, 208
1260-M	Present Tendencies in Exploration for New Mines	S. Locke	San Francisco	69, 3, 14
1267-M	Loading Ore Underground with Scrap- ers at the Utah-Apex Mine	S. P. Holt	Canadian Aug., 1923	69, 362
1268-M	Emergency Power for Mines	G. Bright	Canadian Aug., 1923	69, 369, 373
1271-M	Ground Movement and Subsidence Papers Presented at New York Meet- ing, February, 1923	G. S. Rice, J. Parke Channing, J. J. Rutledge	New York Feb., 1923	69, 374, 394, 406, 414
1287-M	Geology and Mining Methods of Kennecott Mines	S. Birch	New York Feb., 1924	72, 499
1288-M	Geology and Mining Methods of Beat- son Mine	S. Birch	New York Feb., 1924	72, 147, 510
1295-M	Mining Methods of the Telluride Dis- trict	C. N. Bell	New York	72, 550, 564
1309-M	Mining Methods of Verde Mining Dis- trict	C. E. Mills	New York Feb., 1923	72, 381
1310-M	Development of Mine Transportation in Clifton-Morenci District	N. Carmichael, J. Kiddie	New York Feb., 1924	70, 826, 863
1314-M	Mining Methods in Mogollon District, New Mexico	S. J. Kidder	New York Feb., 1924	72, 529
1317-S	Physical Defects in Hollow Drill Steel	F. B. Foley, C. Y. Clayton, M. L. Frey	New York Feb., 1924	70, 290
1329-M	Estimation of Ore Reserves and Mining Methods in Alaska Juneau Mine	P. R. Bradley	New York Feb., 1924	72, 100, 118

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1360-M	Factors Affecting Bank Slopes in Steam-shovel Operations	L. S. Cates	New York Feb., 1925	74, 818, 825
1368-M	Red Iron Ore Mining Methods in the Birmingham District	W. R. Crane	Birmingham Oct., 1924	72, 157
1371-M	Mining Methods at Mascot Mines, Tennessee	H. A. Coy, J. A. Noble	Birmingham Oct., 1924	72, 54
1378-M	Ducktown, Tennessee, Copper District	W. A. Nelson	Birmingham Oct., 1924	71, 299
1379-M	Mascot, Tennessee, Zinc Area	W. A. Nelson	Birmingham Oct., 1924	71, 289
1388-A	Properties of Liquid-oxygen Explosives	G. St. J. Perrott	New York Feb., 1925	71, 1248, 1275
1393-A	Mining Methods in Zaruma District, Ecuador	R. Emmel	New York Feb., 1925	72, 447, 464
1400-A	Mining Methods of Jarbidge District	J. F. Park	New York Feb., 1925	72, 518, 528
1408-A	Mining Methods at the Homestake	A. J. M. Ross, R. G. Wayland	New York Feb., 1925	72, 424, 445
1411-A	Systems of Coal Mining in Western Washington	S. H. Ash	New York Feb., 1925	72, 833, 871
1413-A	Methods of Mining and Ore Estimation at Lucky Tiger Mine	P. T. Mishler, L. R. Budrow	New York Feb., 1925	72, 468, 483
1445-A	Determination of Suspensoids by Alter- nating-current Precipitators	P. Drinker, R. M. Thomson	New York Feb., 1925	71, 1066
1450-A	Discussion of Mining Methods Papers Presented at New York Meeting, 1925			
1466-A	Theoretical Rating Compared with Operating Performance of Centrif- ugal Mine Ventilating Fans	A. S. Richardson	New York Feb., 1926	
1475-A	Mine Subsidence in the Red Iron Ore Mines of the Birmingham District, Alabama	W. R. Crane	New York Feb., 1925	72, 157
1477-A	Discussion of Papers Presented at Ground Movement and Subsidence Sessions, New York Meeting, 1925			
1485-A	Deep-hole Prospecting at the Chief Consolidated Mines	C. A. Dobbel	Salt Lake City Sept., 1925	72, 677
1496-A	Mining Methods in the Mother Lode District of California	S. L. Arnot	Salt Lake City Sept., 1925	
1498-A	Shovel Operations at Bingham, Utah Copper Co.	H. C. Goodrich	Salt Lake City Sept., 1925	72, 566
1499-A	Experiments in Shot-firing with Low- and High-voltage Currents	A. C. Watts	Salt Lake City Sept., 1925	74, 512
1507-A	Top Slicing in Old Fills at El Bordo Mine, Mexico	R. J. Meehin	New York Feb., 1926	72, 139; 74, 687
1512	Discussion of Papers Presented at Salt Lake City, 1925			
1517-A	Subsidence Around a Salt Well	C. M. Young	New York Feb., 1926	74, 810, 816
1525-A	Mining Methods in Grass Valley District, Cal.	J. A. Fulton, A. B. Foote	New York Feb., 1926	74, 654, 664
1542-A	Operating Characteristics of C fugal Fans and the Use of the Fan Performance Curve	L. W. Huber	New York Feb., 1926	

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1559-A	Economic Design of Mine Airways	A. S. Richardson	New York Feb., 1926	74, 342
1562-A	Liquid Oxygen as an Explosive	F. W. O'Neil, H. Van Fleet	New York Feb., 1926	74, 690, 731
1564-A	Discussion of Theory of Mine Ventilation	Sub-committee on Physics of Mine Ventilation	New York Feb., 1926	74, 276
1578-A-F	Application of Kutter's Formula to Gases	F. E. Brackett	Pittsburgh Oct., 1926	74, 312, 321
1580-A-F	Permissible Limits of Toxic and Noxious Gases in Mine and Tunnel Ventilation	R. R. Sayers	New York Feb., 1926	74, 352, 368
1582-A-F	The Holland Tunnel (The Hudson River Vehicular Tunnel)	Ole Singstad	New York Feb., 1926	74, 366, 376
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1606-A-F	Mine Air Flow	G. E. McElroy		74, 297, 305
1626-A	Discussion of Pamphlet No. 1542-A, "Operating Characteristics of Centrifugal Fans and Use of the Fan Performance Curve." Presented at the New York Meeting, February, 1926	L. W. Huber		
1640-A-F	Does the Subsurface Owner Owe an Absolute Duty to Support the Surface?	R. G. Bosworth	New York Feb., 1927	
1641-A	Acceleration Stresses in Hoisting Cables	G. P. Boomslicer	New York Feb., 1927	75, 74, 95
1642-A-F	Mine Timber Preservation	G. M. Hunt	New York Feb., 1927	
1649-A	Underground Air Conditions and Ventilation Methods at Tonapah, Nevada	B. O. Pickard	New York Feb., 1927	75, 107, 120
1652-A-F	Liquid Oxygen Explosives in Strip Coal Mining	G. B. Holderer	New York Feb., 1927	
13-A-1	Ore at Deep Levels in the Cripple Creek District, Colorado	G. F. Loughlin	Denver Sept., 1926	75, 42
17-A-2 G-1	Subsidence, and Earth Movements Caused by Oil Extraction or by Drilling Oil and Gas Wells	W. T. Thom, Jr.	New York Feb., 1927	75, 734, 737
20-A-3	The Latouche System of Mining as Developed at the Beatson Mine, Kennecott Copper Corporation Latouche, Alaska	B. Presley	New York Feb., 1928	
21-A-4	Ventilation of the Liberty Tunnels	L. W. Huber	New York Feb., 1928	75, 122, 132
45-A-5 I-1	Geophysical Exploration for Ores	Max Mason	N. Y. Sect. Oct., 1927	

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1028	Electrolytic Zinc Plant of Anaconda Copper Mining Co.	F. Laist, F. F. Frick, J. O. Elton, R. B. Caples	New York Feb., 1921	64, 699, 753
1042	Important Factors in Talc Milling Efficiency	R. B. Ladoo	New York Feb., 1921	66, 82, 87
1085-M	Flotation of Pyrite	W. S. Morley	Wilkes-Barre Sept., 1921	

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1185-M	Surface Tension and Adsorption Phenomena in Flotation	A. F. Taggart, A. M. Gaudin	San Francisco Sept., 1922	68, 479, 530
1212-M	Heap Leaching at Bisbee, Arizona	A. W. Hudson G. D. Van Arsdale	New York Feb., 1923	69, 137, 152
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1262-M	A Contribution to the Kick versus Rittinger Dispute	H. E. T. Haultain	Canadian Aug., 1923	69, 183, 188
1283-M	Surface Reactions in Flotation	A. W. Fahrenwald	New York Feb., 1924	70, 647, 737
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1293-M	Efficiency of Screening	R. K. Warner	New York Feb., 1924	70, 631, 640
1297-M	Ammonia Leaching of Calumet and Hecla Tailings	C. H. Benedict, H. C. Kenny	New York Feb., 1924	70, 595, 610
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1361-M	Determination of Dissolved Oxygen in Cyanide Solutions	A. J. Weinig, M. W. Bowen	New York Feb., 1925	71, 1018, 1086
1362-D	A New Roasting Furnace for Zinc Flotation Concentrate	C. H. Fulton, J. Burns Read	New York Feb., 1925	71, 929, 950
1372-M	Milling Practice of American Zinc Co. of Tennessee at Mascot	R. Ammon	Birmingham Oct., 1924	71, 1029
1386-B	Recent Developments in Fine Grinding and Treatment of Witwatersrand Ores	C. R. Davis, J. L. Willey, S. E. T. Ewing	New York Feb., 1925	71, 983, 1014
1407-B	Precipitation Efficiency of Zinc Dust in Cyanide Process	R. Lepsoe	New York Feb., 1925	71, 1061, 1085
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1481-B	Chloridizing Mill of the Standard Reduction Co.	H. P. Allen, Wm. C. Madge	Salt Lake City Sept., 1925	73, 317, 333
1482-B	Method of Unloading Ores and Coarse-crushing Practice at Magna Plant of Utah Copper Co.	B. E. Mix, L. M. Barker	Salt Lake City Sept., 1925	73, 212, 224
1484-B	Milling Practice at Midvale	C. A. Lemke	Salt Lake City Sept., 1925	73, 342
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1541-B	Mechanism of Filtration	A. W. Hixon, L. T. Work, I. H. Odell	New York Feb., 1926	73, 225, 234
1560-B	Classification in Witwatersrand Mills	B. R. Bates	New York Feb., 1926	73, 239, 249
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1616-B	Degree of Liberation of Minerals in the Alabama Low-grade Red Iron after Grinding	W. H. Coghill	New York Feb., 1927	75, 147, 160
1653-B	Beneficiation of Alabama Siliceous Red Hematite	J. T. Singewald, Jr.	New York Feb., 1927	
1-B-1	Flotation Practice in the Coeur d'Alene District, Idaho	A. W. Fahrenwald	Salt Lake City Aug., 1927	
3-B-2	Effect of Preferential Flotation at Cananea Mill and Smelter	A. T. Tye	Salt Lake City Aug., 1927	
4-B-3	Flotation Mechanism, A Discussion of the Functions of Flotation Reagents	A. M. Gaudin	Salt Lake City Aug., 1927	
5-B-4	Recent Flotation Practice at Inspiration, Arizona	Guy H. Ruggles, H. F. Adams	Salt Lake City Aug., 1927	
6-B-5	The Forrester Cell Installation at the Nevada Consolidated Copper Co.'s McGill Concentrator	E. H. Mohr	Salt Lake City Aug., 1927	
7-B-6	Concentration of Lead-silver Ore at Hecla Mine, Gem, Idaho	W. L. Zeigler	Salt Lake City Aug., 1927	
8-B-7	Galena Flotation Concentrator, Lake Gulch, Idaho	W. L. Zeigler	Salt Lake City Aug., 1927	
9-B-8	Ore Concentration Practice of the Consolidated Mining & Smelting Co. of Canada, Ltd.	R. W. Diamond	Salt Lake City Aug., 1927	
10-B-9	Flotation of Oxidized Lead-silver Ores	A. W. Hahn	Salt Lake City Aug., 1927	

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1029	Steel Chimneys and Their Linings in Copper Smelting Plants	A. G. McGregor	New York Feb., 1921	64, 830, 836
1030	Importance of Hardness of Blast-furnace Coke	Owen R. Rice	New York Feb., 1921	67, 596, 604
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1044	The Electric Furnace in the Iron Foundry	R. Moldenke	New York Feb., 1921	67, 544, 551
1046	Discussion of Papers Presented at New York Meeting, 1921			
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1051	Chemical Equilibrium between Iron, Carbon and Oxygen	A. Matsubara	New York Feb., 1921	67, 3, 54
1052	Measurement of Blast-furnace Gas	D. L. Ward, R. S. Reed	New York Feb., 1921	67, 624, 628
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1071	Thacher Molding Process for Propeller Wheels and Blades	E. Touceda	Wilkes-Barre Sept., 1921	67, 532
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1080	What is the Ideal Drill Steel?	F. H. Kingdon	New York Feb., 1921	66, 807
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1462-C	Discussion of Iron and Steel Papers Presented at New York Meeting, 1925			
1470-C	Economic Significance of Cyanide Accumulation in the Blast Furnace	R. Franchot	New York Feb., 1926	
1478	Discussion of Papers Presented at Birmingham, 1924			
1512	Discussion of Papers Presented at Salt Lake City, 1925			
1518-C	Effect of Air Gap in Explosion System on Production of Neumann Bands	F. B. Foley, J. E. Crawshaw	New York Feb., 1926	73, 948, 964
1532-C	Current Theories of the Hardening of Steel Thirty Years Later	A. Sauveur	New York Feb., 1926	73, 859, 902
1545-C	Influence of Temperature, Time and Rate of Cooling on Physical Properties of Carbon Steel	F. B. Foley, C. Y. Clayton, W. E. Remmers	New York Feb., 1926	73, 929
1550-C	Elimination of Metalloids in the Basic Open-hearth Process	J. L. Keats, C. H. Herty, Jr.	New York Feb., 1926	73, 1079
1556-A	Sharpening and Handling Drill Steels at Franklin, N. J.	C. M. Haight	New York Feb., 1926	74, 666, 679
1558-C	Effect of Annealing upon the Hardness of Cold-worked Ingot Iron	C. Y. Clayton	New York Feb., 1926	73, 926
1563-C	Making Rimmed Steel	C. Pierce	New York Feb., 1926	73, 1026, 1036
1566-C	Chemical Equilibrium of Manganese, Carbon and Phosphorus in the Basic Open-hearth Process	C. H. Herty, Jr.	New York Feb., 1926	73, 1107
1567-C	Discussion of Papers Presented at Iron and Steel Open-hearth Session at New York Meeting, 1926			
1568-C	Combustion in the Open-hearth Furnace with Special Reference to Automatic Control	K. Huessener	New York Feb., 1926	73, 1047
1572-C	A Process for the Prevention of Embrittlement in Malleable Cast Iron	L. H. Marshall	New York Feb., 1926	
1576-C	Introduction to Ultra-violet Metallography	F. F. Lucas	New York Feb., 1926	73, 909, 925
1579-C	Progress Report on the Effect of the Open-hearth Process on Refractories	F. W. Schroeder, B. M. Larsen	Pittsburgh Oct., 1926	

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PAMPHLET NUMBER	TITLE	AUTHOR	MEETING	TRANS. VOLUME AND PAGE
1581-C	Optical Temperature Measurements in Open-hearth Furnaces	B. M. Larsen, J. W. Campbell	Pittsburgh Oct., 1926	75, 245
1591-C	The Manufacture of Forging Steel by the Basic Open-hearth Process	R. L. Cain	Pittsburgh Oct., 1926	
1595-C	The Trend of Development in the Wrought Iron Industry	J. Aston	Pittsburgh Oct., 1926	
1596-C	Specific Efficiency of the Blast Furnace	R. Franchot	Pittsburgh Oct., 1926	
1597-C	The Desulfurizing Action of Manganese in Iron	C. H. Herty, Jr., J. M. Gaines, Jr.	Pittsburgh Oct., 1926	75, 434, 439
1600-C	Bend Tests of Galvanized Sheet Steel	H. A. Stacy	Pittsburgh Oct., 1926	
1603-C	Composition of Iron Blast Furnace Slags	R. S. McCaffery, J. F. Osterle, Leo Schapiro	Pittsburgh Oct., 1926	
1612-C	Nature of the Chromium-iron-carbon Diagram	M. A. Grossmann	New York Feb., 1927	75, 214, 233
1622-C-I	Geology of the Manganese Ore Deposits of the Gold Coast, Africa With Notes on the Petrology of Certain Associated Manganese Silicate-bearing Rocks	A. E. Kitson, N. R. Junner	Cleveland Apr., 1927	75, 372, 395
1648-C	Need for Research in Foundry Pig Iron	R. Moldenke	New York Apr., 1927	75, 443, 447
1650-C	An Introduction to the Iron-chromium-nickel Alloys	E. C. Bain, W. E. Griffiths	New York Apr., 1927	75, 166, 211
1655-C-E	Manganese in Non-ferrous Alloys	M. G. Corson	Cleveland Apr., 1927	
1656-C	Manganese Resources in Relation to Domestic Consumption	J. V. W. Reynders	Cleveland Apr., 1927	75, 272, 285
1659-C	A Theory of the Cause of Blisters on Galvanized Sheets	L. B. Lindemuth	New York Feb., 1927	
1660-C	The Importance of Manganese in the Steel Industry	H. M. Boylston	Cleveland Apr., 1927	75, 397, 408
1662-C	A Comparison of the Effect of Nickel and Cobalt in Steel	F. H. Allison, Jr.	New York Feb., 1927	75, 234
1664-C	Reserves of Lake Superior Manganiferous Iron Ores, with Appendix to the Paper	C. Zapfe	Cleveland Apr., 1927	75, 346, 369
1665-C	Minnesota Manganiferous Iron Ores in Relation to the Iron and Steel Industry	T. L. Joseph, E. P. Barrett, C. E. Wood	Cleveland Apr., 1927	75, 292, 338
1666-C	Discussion of Paper (Pamphlet No. 1656-C), Manganese Resources in Relation to Domestic Consumption			
11-C-1	Round Table: Carbon in Pig Iron	R. Moldenke, W. B. Coleman, E. Touceda, R. H. Sweetser	New York Feb., 1927	75, 443, 457, 469, 473, 447, 468, 471, 480, 488
14-C-2	"Heterogeneity of Iron-manganese Alloys	C. R. Wöhrman	New York Feb., 1928	

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1028	Electrolytic Zinc Plant of Anaconda Copper Mining Co.	F. Laist, F. F. Frick, J. O. Elton, R. B. Caples	New York Feb., 1921	64, 699, 753
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1046	Discussion of Papers Presented at New York Meeting, 1921			
1073	Discussion of Papers Presented at New York Meeting, 1921			
1079-N	Queen Nine-hearth Roaster	J. M. Samuel	Wilkes-Barre Sept., 1921	
1108	Discussion of Papers at New York and Wilkes-Barre, 1921			
1136-M	Arsenical Bearing Metals	H. J. Roast, C. F. Pascoe	New York Feb., 1922	68, 735, 744
1160-N	Gas Absorption and Oxidation of Non-ferrous Metals	B. Woyaki, J. W. Boeck	Rochester, June, 1922	68, 861, 868
1197-M	Soluble Silica in the Preparation of Zinc-sulfate Solution for Electrolysis	J. O. Betterton	New York Feb., 1923	69, 159, 167
1198-N	Occurrence, Chemistry and Uses of Selenium and Tellurium	V. Lenher	New York Feb., 1923	69, 1035, 1055
1200	Discussion of Papers Presented at San Francisco, 1922			
1202-N	Colloid Chemistry and Metallurgy	W. D. Bancroft	New York Feb., 1922	68, 804, 1152
1204-N	Pyritic Smelting and Basic Converting at the Kosaka Copper Smelter, Japan	Kenzo Ikeda	New York Feb., 1923	69, 123, 130
1273-M	Direct Electrolysis of Black-copper Anodes of High Nickel-lead Content	M. H. Merriss	New York Feb., 1924	70, 581, 593
1274-M	Calculating the Zinc for Desilverizing Lead Bullion by the Parkes Process	G. G. Griswold	New York Feb., 1924	70, 611, 614
1281-M	Hydrometallurgy of Lead	O. C. Ralston	New York Feb., 1924	70, 447, 466
1284-M	Greenawalt Electrolytic Copper Extraction Process	W. E. Greenawalt	New York Feb., 1924	70, 529, 567
1301-M	Electrolytic Zinc from Complex Ores	U. C. Tainton, L. T. Leyson	New York Feb., 1924	70, 486, 523
1302-M	Effect of Oxygen-enriched Air in Roasting Zinc Ores	B. M. O'Harra, W. Kahlbaum, E. S. Wheeler, W. J. Darby	New York Feb., 1924	
1311-M	Present Trend in Treatment of Complex Ores	G. L. Oldright	New York Feb., 1924	70, 471
1331-M	Reduction and Refining of Tin in the United States	H. H. Alexander, J. R. Stack	New York Feb., 1924	70, 404, 440
1344-M	Discussion of Metallurgical Papers Presented at New York Meeting, 1924			
1349-M	Redistillation of Zinc	Kurt Stock	New York Feb., 1925	71, 897, 918
1366-M	Smelting Copper Concentrates in a Converter	F. J. Longworth	Birmingham Oct., 1924	71, 969
1377-S	Use of Oxygenated Air in Metallurgical Operations	Abstracts of Symposium Papers	New York Feb., 1924	
1406-D	High Zinc in Lead Blast-furnace Slags	F. E. Beasley	New York Feb., 1925	71, 919, 921
1425-D	Application of Pulverized Coal to Copper Refinery Furnaces	E. W. Steele	New York Feb., 1925	71, 972, 982
1452-D	Recovery of Arsenic and Other Valuable Constituents from Speiss	C. P. Linville	New York Feb., 1925	71, 953, 965
1455-D	Discussion of Non-ferrous Metallurgical Papers Presented at New York Meeting, 1925			
1467-D	Improvements in the Series System of Electrolytic Copper Refining Recently Developed by the Nichols Copper Co.	M. H. Merriss	New York Feb., 1926	73, 123, 142

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PAMPHLET NUMBER	TITLE	AUTHOR	MEETING	TRANS. VOLUME AND PAGE
1474-D	Effect of Zinc Oxide on the Formation Temperatures of Some Ferrous Slags	H. Tharp Mann	New York Feb., 1926	73, 3, 28
1480-D	Application of Cottrell Process in Lead and Copper Smelting	A. L. Labbe	Salt Lake City Sept., 1925 "	
1483-D	Evolution in the Preparation of Ores for Lead Blast Furnaces	D. W. Jessup	Salt Lake City Sept., 1925	
1486-D	Lead Smelting in Utah	B. L. Sackett, C. Bardwell, S. Jacobson, N. H. Jensen	Salt Lake City Sept., 1925	73, 171, 197
1489-D	Production of Ferric Sulfate and Sul- furic Acid from Roaster Gas	G. L. Oldright, H. E. Keyes, F. S. Wartman	Salt Lake City Sept., 1925	73, 84, 98
1492-D	Recovery of Copper by Leaching, Ohio Copper Co. of Utah	A. E. Anderson, F. K. Cameron	Salt Lake City Sept., 1925	73, 31, 56
1497-D	Pulverized Coal as Fuel for Copper- refining Furnaces	E. S. Bardwell, R. H. Miller	Salt Lake City Sept., 1925	74, 592
1502-D	Leaching Mixed Copper Ores with Fer- ric Sulfate; Inspiration Copper Co.	G. D. VanArsdale	New York Feb., 1926	73, 58, 74
1504-D	Anaconda Electrolytic White Lead	R. G. Bowman	Salt Lake City Sept., 1925	73, 146
1512	Discussion of Papers Presented at Salt Lake City, 1925			
1537-D	Endurance Properties of Non-ferrous Metals	D. J. McAdam, Jr.	New York Feb., 1926	
1547-D	Relation between Metallurgy and Atomic Structure	P. D. Foote	New York Feb., 1926	73, 628
1548-D	Effect of Lead and Tin with Oxygen on the Conductivity and Ductility of Copper	N. B. Pilling, G. P. Halliwell	New York Feb., 1926	73, 679, 692
1549-D	Treatment of the Telluride-bearing Gold Ores of the Wright-Hargreaves Mines, Ltd.	W. A. Mueller, J. E. Grant, C. L. Heath	New York Feb., 1926	
1574-D	Conductivity of Electrolytes Used in the Electrolytic Separation of Silver and Gold	F. F. Colcord, E. F. Kern, J. J. Mulligan	New York Feb., 1926	73, 108, 117

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1031	Nickel-chromium Alloys	Leon O. Hart	Columbus Oct., 1920	64, 554
1032	Effect of Temperate, Deformation, Grain Size and Rate of Loading on Mechanical Properties of Metals	W. P. Sykes	New York Feb., 1921	64, 780, 814
1038	Discussion of Papers Presented in 1920			
1073	Discussion of Papers Presented at New York Meeting, 1921			
1075-M	Investigation of Fatigue of Metals under Stress	H. F. Moore	New York Feb., 1921	66, 806
1087-S	Making a 5-per cent. Nickel-cast-iron Alloy in an Electric Furnace	D. N. Witman	Wilkes-Barre Sept., 1921	67, 523
1104-N	Cracks in Aluminum-alloy Castings	R. J. Anderson	New York Feb., 1922	68, 833, 854
1131-N	Experiments with Sherardizing	L. McCulloch	New York Feb., 1922	68, 757, 760
1133-N	Studies on the Constitution of Binary Zinc-base Alloys	W. M. Peirce	New York Feb., 1922	68, 767, 792
1134-N	Spectrum Analysis in an Industrial Laboratory	W. H. Bassett, C. H. Davis	New York Feb., 1922	68, 862, 867

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1139-N	Crystal Structure of Solid Solutions	E. C. Bain	New York Feb., 1922	68, 625, 639
1151-N	Testing Artillery Cartridge Cases	J. Burns Read, S. Tour	New York Feb., 1921	68, 670, 723
1152-N	Core-oven Tests	F. L. Wolf, A. A. Grubb	Rochester June, 1922	68, 878, 890
1158-N	Occurrence of Blue Constituent in High-strength Manganese Bronze	E. H. Dix, Jr.	Rochester June, 1922	68, 642, 657
1159-N	Some Electrical Properties of Nickel and Monel Wires	M. A. Hunter, F. M. Sebast, A. Jones	Rochester, June, 1922	68, 750
1161-N	Physical Properties of Cartridge Brass	C. Upthegrove, W. G. Harbert	Rochester June, 1922	68, 725, 734
1163	Discussion of Institute of Metals and Iron and Steel Papers Presented at New York Meeting, 1922			
1181-S	Conditions of Stable Equilibrium in Iron-carbon Alloys	H. A. Schwartz, H. R. Payne, A. F. Gorton, M. M. Austin	San Francisco Sept., 1922	68, 916, 927
1184-N	Effect of Impurities on the Oxidation and Swelling of Zinc Aluminum Alloys	H. E. Brauer, W. M. Peirce		68, 796, 827
1186-S	Neumann Bands as Evidence of Action of Explosives upon Metal	F. B. Foley, S. P. Howell	San Francisco Sept., 1922	68, 891, 915
1190	Discussion of Papers Presented at Meeting of Institute of Metals Division, June, 1922			
1202-N	Colloid Chemistry and Metallurgy	W. D. Bancroft	New York Feb., 1922	68, 604, 1158
1213-N	Observations on the Occurrence of Iron and Silicon in Aluminum	E. H. Dix, Jr.	New York Feb., 1923	69, 957, 971
1215-N	Bright Annealing of Copper Wire in an Atmosphere of Natural Gas	P. E. Demmler	New York Feb., 1923	69, 1058, 1061
1216-N	A Study of Bearing Metals	C. H. Bierbaum	New York Feb., 1923	69, 972, 983
1226-S	Influence of Temperature, Time and Rate of Cooling on Physical Properties of Carbon Steel	H. M. Howe, F. B. Foley, J. Winlock	New York Feb., 1923	69, 722
1229-N	Thermal Conductivity of Some Industrial Alloys	H. M. Williams, V. W. Bihlman	New York Feb., 1923	69, 1065
1249-N	Discussion of Institute of Metals Division Papers Presented at New York Meeting, 1923			
1250-N	Solid Solutions	W. Rosenhain	New York Feb., 1923	69, 1003
1257-S	Selecting Material for Formed and Drawn Parts	L. N. Brown	Canadian Aug., 1923	69, 932, 941
1259-N	Effect of Heat Treatment on Release of Stress in Bronze Castings	R. J. Anderson, C. H. Eldridge	Cleveland Apr., 1923	69, 990, 1001
1263-S	X-ray Examination of Irregular Metal Objects	Ansel St. John	Canadian Aug., 1923	69, 943, 953
1264-S	Effect of Silicon on Equilibrium Diagram of System Carbon-iron near Eutectoid Points	H. A. Schwartz, H. R. Payne, A. F. Gorton	Canadian Aug., 1923	69, 791, 796
1265-S	Some Commercial Alloys of Iron, Chromium and Carbon in the Higher Chromium Ranges	C. E. MacQuigg	Canadian Aug., 1923	69, 831
1266-S	Some Effects of Zirconium in Steel	A. L. Feld	Canadian Aug., 1923	69, 848, 853

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1278-S	Overstrain in Metals	J. K. Wood	New York Feb., 1924	70, 274
1282-N	Corrosion of Brass as Affected by Grain Size	R. J. Anderson, G. M. Enos	New York Feb., 1924	70, 391, 401
1289-S	Micrographic Detection of Carbides in Ferrous Alloys	N. B. Pilling	New York Feb., 1924	70, 254
1291-N	Effect of Severe Cold Working on Scratch and Brinell Hardness	H. S. Rawdon, W. H. Mutchler	New York Feb., 1924	70, 342, 352
1303-N	Some Low Copper-nickel Silvers	Wm. B. Price, C. G. Grant	New York Feb., 1924	70, 328
1304-S	Use of Sodium Picrate in Revealing Dendritic Segregation in Iron Alloys	A. Sauveur, V. N. Krivobok	New York Feb., 1924	70, 239, 251
1305-N	Relation of Heat Treatment to the Microstructure of 60-40 Brass	R. S. Williams, V. O. Homerberg	New York Feb., 1924	70, 375, 389
1333-S	Simulating Natural Light in Metallog- raphy	H. S. George	New York Feb., 1924	70, 259
1334-N	Relation of Heat Treatment, Mechani- cal Properties and Microstructure of 60-40 Brass	V. O. Homerberg, D. N. Shaw	New York Feb., 1924	70, 365
1337-N	The Trend in the Science of Metals	Zay Jeffries	New York Feb., 1924	70, 303
1340-N	Discussion of Papers of Institute of Metals Division Presented at New York Meeting, 1924			
1348-N	Experiments on the Heat Treatment of Alpha-beta Brass	O. W. Ellis, D. A. Schemnitz	Milwaukee Oct., 1924	71, 794, 805
1350-N	Casting and Heat Treatment of Some Aluminum-copper-magnesium Alloys	S. Daniels, A. J. Lyon, J. B. Johnson	Milwaukee Oct., 1924	71, 864, 885
1357-N	Hardness of Heat-treated Aluminum Bronze	G. F. Comstock	Milwaukee Oct., 1924	71, 806, 814
1358-N	Coatings Formed on Corroded Metals and Alloys	G. M. Enos, R. J. Anderson	Milwaukee Oct., 1924	71, 784, 792
1382-E	Recrystallization and Grain Growth in Soft Metals	M. Cook, U. R. Evans	New York Feb., 1925	71, 627, 650
1387-E	Determination of Structural Compo- sition of Alloys by a Metallographic Planimeter	E. P. Polushkin	New York Feb., 1925	71, 669, 687
1392-E	X-ray Evidence versus the Amor- phous-metal Hypothesis	R. J. Anderson, J. T. Norton	New York Feb., 1925	71, 720, 732
1394-E	Corrosion of Copper Alloys in Sea Water	W. H. Bassett, C. H. Davis	New York Feb., 1925	71, 745, 774
1395-E	Scratch and Brinell Hardness of Severely Cold-rolled Metals	M. F. Fogler, E. J. Quinn	New York Feb., 1925	71, 889, 894
1398-E	The Malleability of Nickel	P. D. Merica, R. G. Waltenberg	New York Feb., 1925	71, 709, 716
1404-E	Discussion of Papers of Institute of Metals Division Presented at Mil- waukee Meeting, 1924			
1415-E	New Developments in High-strength Aluminum Alloys	R. S. Archer, Zay Jeffries	New York Feb., 1925	71, 828, 845
1418-E	Etching Aluminum and Its Alloys for Macroscopic and Microscopic Exami- nation	F. B. Flick	New York Feb., 1925	71, 816, 827
1421-E	Eutectic Patterns in Metallic Alloys	C. H. Green	New York Feb., 1925	71, 651, 665
1426-E	Tantalum, Tungsten and Molybdenum	E. W. Engle	New York Feb., 1925	71, 691
1449-E	Action of Hot Wall: a Factor of Fundamental Influence on the Rapid Corrosion of Water Tubes and Related to the Segregation in Hot Metals	C. Benedicks	New York Feb., 1925	71, 597

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PAMPHLET NUMBER	TITLE	AUTHOR	MEETING	TRANS. VOLUME AND PAGE
1454-E	Discussion of Papers of Institute of Metals Division Presented at New York Meeting, 1925			
1464-E	Mechanical Properties of the Alumi-num-copper-silicon Alloy as Sand Cast and as Heat Treated	S. Daniels, D. M. Warner	Syracuse Oct., 1925	73, 464
1465-E	Cracking of the Nickel Silvers in the Course of Annealing	E. O. Jones, E. Whitehead	Syracuse Oct., 1925	73, 834, 849
1505-C	Amorphous Cement and the Formation of Ferrite in the Light of X-ray Evidence	F. B. Foley	Syracuse Oct., 1925	73, 850
1506-E	Endurance Properties of Non-ferrous Metals	D. J. McAdam, Jr.	Syracuse Oct., 1925	
1508-E	Special Nickel Brasses	O. Smalley	Syracuse Oct., 1925	73, 799
1509-E	The Microstructure of Aluminum	K. L. Meissner	New York Feb., 1926	73, 622, 626
1510-E	Discussion of Papers Presented at Institute of Metals Meeting in Syracuse, 1925			
1514-E	Some Examples of Copper Made Brittle by Hot Reducing Gases	T. S. Fuller	New York Feb., 1926	73, 744, 750
1515-C	The Iron-tungsten System	W. P. Sykes	New York Feb., 1926	73, 968, 1004
1518-C	Effect of Air Gap in Explosion System on Production of Neumann Bands	F. B. Foley, J. E. Crawshaw	New York Feb., 1926	73, 948, 964
1519-E	Effect of Reheating on the Al-Cu-Ni-Mg and the Al-Cu-Fe-Mg (Piston) Alloys	S. Daniels	New York Feb., 1926	73, 479
1520-E	Exudations on Copper Castings	W. H. Bassett, J. C. Bradley	New York Feb., 1926	73, 700, 706
1521-E	Action of Reducing Gases on Heated Copper	W. H. Bassett, J. C. Bradley	New York Feb., 1926	73, 755, 770
1523-E	Annealing of Commercial Copper to Prevent Embrittlement of Reducing Gases	S. B. Leiter	New York Feb., 1926	73, 776, 782
1524-E	Microscopic Structure of Copper	H. B. Pulsifer	New York Feb., 1926	73, 707, 739
1526-E	Estimation of Oxygen and Sulfur in Refined Copper	W. H. Bassett, H. A. Bedworth	New York Feb., 1926	73, 784, 797
1527-E	Hardness of Copper, and Meyer's Analysis	S. L. Hoyt, T. R. Schermerhorn	New York Feb., 1926	73, 657, 671
1534-E	Equilibrium Relations in Aluminum-copper Alloys of High Purity	E. H. Dix, Jr., H. H. Richardson	New York Feb., 1926	73, 560
1538-E	A Preliminary Study of Magnesium Base Alloys	B. Stoughton, M. Miyake	New York Feb., 1926	73, 541, 557
1539-E	The Lead-antimony System and the Hardening of Lead Alloys	R. S. Dean, L. Zickrick, F. C. Nix	New York Feb., 1926	73, 505, 529
1544-E	Modification and Properties of Sand-cast Aluminum-silicon Alloys	R. S. Archer, L. W. Kempf	New York Feb., 1926	73, 581, 619
1557-E	A Photomicrographic Study of the Process of Recrystallization in Certain Cold Worked Metals	V. N. Krivobok	New York Feb., 1926	
1586-E	Exudations on Brass and Bronze	W. B. Price, A. J. Phillips	Detroit Sept., 1926	M-27,* 80, 88
1588-E	Constitution of Alloys of Aluminum, Zinc and Tin and Aluminum, Zinc and Cadmium	V. Jareš	Detroit Sept., 1926	M-27, 65

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1590-E	Aluminum Castings of High Strength	R. S. Archer, Zay Jeffries	Detroit Sept., 1926	M-27, 35, 80
1610-E	Plastic Deformation of Metals	J. T. Norton, B. E. Warren	New York Feb., 1927	M-27, 350, 367
1613-E	Note on the Distribution of Energy in Worked Metals and the Effect of Process Annealing Temperature on the Final Annealing Temperature of Fine Copper Wire	L. Zickrick, R. S. Dean	New York Feb., 1927	M-27, 207, 212
1614-E	Tungsten and Thoria	Zay Jeffries, P. Tarasov	New York Feb., 1927	M-27, 395, 411
1623-E	Plastic Deformation of Coarse-grained Zinc	C. H. Mathewson, A. Phillips	New York Feb., 1927	M-27, 143, 189
1624-E	Plastic Deformation of a Zinc Single Crystal	S. L. Hoyt	New York Feb., 1927	M-27, 116, 132
1630-E	A Study of the Heat Treatment, Micro- structure and Hardness of 60:40 Brass	F. H. Clark	New York Feb., 1927	M-27, 276, 299
1631-E	Preparation of Metallic Single Crystals and Twinning in Zinc and Zinc Single Crystals	O. E. Romig	New York Feb., 1927	M-27, 90, 112
1632-E	Corrosion and Physical Properties of Some Alloys of Aluminum, Zinc and Tin	N. O. Taylor	New York Feb., 1927	M-27, 302, 312
1633-E	Equilibrium Relations in Al-Mn Alloys of High Purity	E. H. Dix, W. D. Keith	New York Feb., 1927	M-27, 315, 333
1634-E	Beryllium-copper Alloys	W. H. Bassett	New York Feb., 1927	M-27, 218, 232
1635-E	Grain Boundary Phenomena in Tung- sten Filaments	E. S. Davenport	New York Feb., 1927	M-27, 413, 427
1636-E	Solidus Line in the Lead-antimony Sys- tem	E. E. Schumacher, F. C. Nix	New York Feb., 1927	M-27, 195, 206
1637-E	A Study of the 470° C. Transition Point in Cast 60:40 Brass	F. H. Clark	New York Feb., 1927	M-27, 253, 270
1643-E	Some Comparative Properties of Tough Pitch and Phosphorized Copper	W. R. Webster, J. L. Christie, R. S. Pratt	New York Feb., 1927	M-27, 233, 249
1644-E	General Theory of Metallic Hardening	R. S. Dean, J. L. Gregg	New York Feb., 1927	M-27, 368, 386
1645-E	Deformation of an Aluminum Alloy by a Constant Load	C. B. Sadtler, J. L. Gregg	New York Feb., 1927	M-27, 336, 348
1646-E	Relation of Annealing Temperature to Conductivity of Copper Wire	J. C. Bradley	New York Feb., 1927	M-27, 210, 212 75, 548, 550
1651-E	Still Casting of Metals	P. H. G. Durville	New York Feb., 1927	M-27, 343, 348
1654-E	Application of Microtome Methods to the Preparation of Soft Metals for Microscopic Examination	F. F. Lucas	New York Feb., 1927	M-27, 481, 495
1655-C-E	Manganese in Non-ferrous Alloys	M. G. Corson	Cleveland Apr., 1927	
1657-E	Notes on the Atomic Behavior of Hard- enable Copper Alloys	E. C. Bain	New York Feb., 1927	M-27, 451, 468
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